Metallurgical and Chemical Engineering

An Incorporation of

Electrochemical and Metallurgical Industry—Iron and Steel Magazine

Vol. XI. No. 12 New York, December, 1913. { 25 cts. per Copy. $2.00 per Year to U. S. and Mexico. $2.50 per Year to other Countries.

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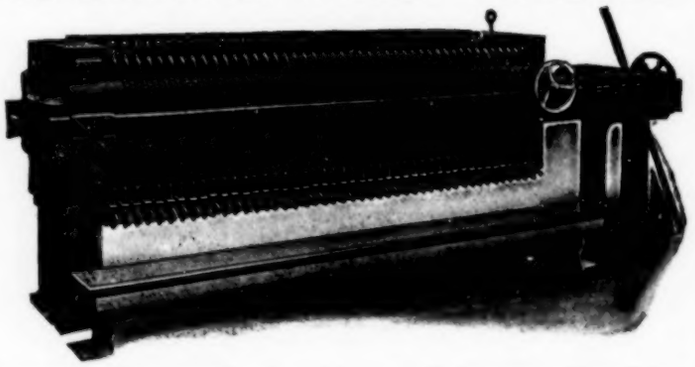

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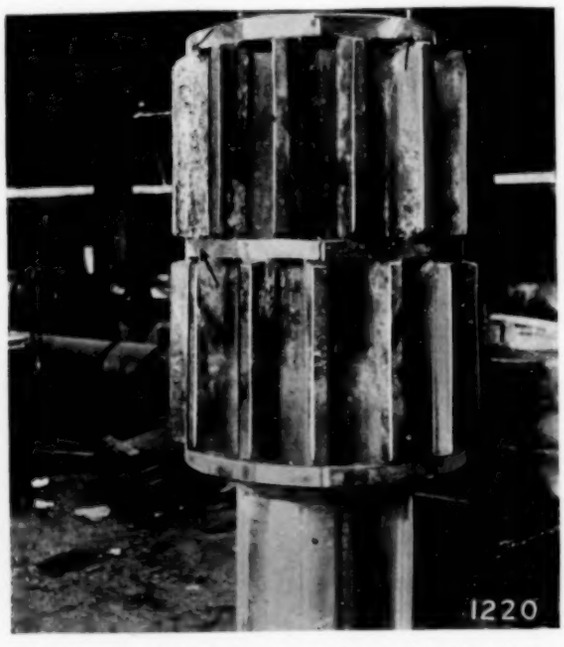

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METALLURGICAL & CHEMICAL ENGINEERING

ISSUE OF DECEMBER, 1913

Electric Furnaces, Their Design, Characteristics, and Commercial Applications. By Woolsey McA. Johnson and Geo. N. Sieger 679
In the present instalment of their illustrated serial the authors discuss the design of buildings for electric-furnace plants, foundations for the furnace, power facilities and voltage regulation, the essentials of electric-furnace transformers and the design of water rheostats.

The Iron-Blast Furnace and the Characteristics of Its Fuels. By J. E. Johnson, Jr 687
The first article of a serial on the iron-blast furnace. After a concise definition of the essential features of the blast-furnace the author discusses the characteristics of blast-furnace fuels and deals with the manufacture and uses as fuel of coke, charcoal, anthracite coal, and raw coal.

The Art of Enameling, or the Coating of Steel and Iron with Glass. By Raymond F. Nailler 695
An article describing the different steps in the art of enameling and giving formulas for various satisfactory mixtures.

Investigations of Blast-Furnace Operations. By W. Mathesius 699
An analysis of the different reactions in the blast furnace with reference to the factors which influence coke consumption, especially the use of sintered or briquetted fuel.

Ozone. By A. Vosmaer 705
In the present instalment of his serial the author, who is an expert on the manufacture of ozone, discusses the manufacture and properties of ozone and its uses in daily life and in the industries.

Novel Electric Steel-Casting Plant. By Ernst M. Schmelz 709
An illustrated description of a new electric steel-casting plant at Redondo, California. This is the first installation of an electric furnace of the pure Stassano type in the United States.

A New Type of Electric Furnace for the Reduction of Ores. By Francis Louvrier 710
An illustrated article on a resistance furnace which permits easy regulation of temperature and its application in the reduction of ores of various metals, especially zinc.

Present Status and Future Prospects of the Metallurgy of Zinc 677
An account of a symposium of papers delivered at the recent joint meeting of the New York Sections of the American Institute of Mining Engineers and the American Electrochemical Society. Geo. C. Stone discussed the development of the retort process, W. R. Ingalls the possibilities of electric zinc smelting, and J. W. Richards the status of electrolytic zinc. An account of the discussion which followed is also given.

The Passivity of Metals 679
An account of a symposium of papers presented before the Faraday Society. G. Senter gave a general introduction into the theory of the passivity of metals, G. Grube discussed anodic and cathodic retardation phenomena, D. Reichinstein discussed the electrode-volume constant-sum hypothesis, H. S. Allen spoke on photo-electricity and passivity, M. LeBlanc on passivity and reaction velocities, G. Schulze on passivity and electrolytic valve action, G. C. Schmidt discussed the hydrogen theory of passivity, and E. P. Schoch the oxygen film or oxide film hypothesis.

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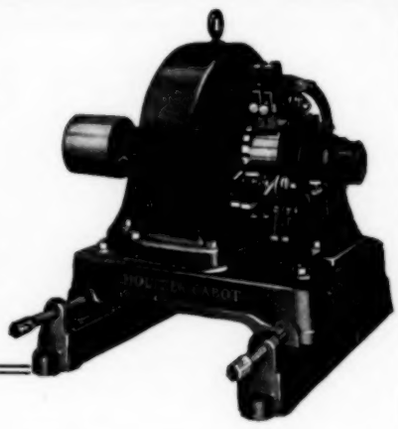

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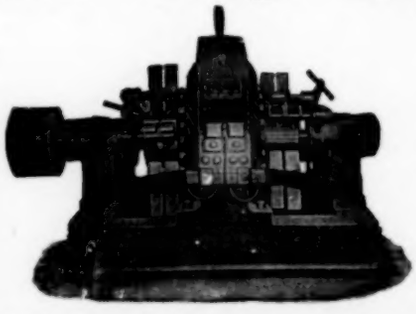

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Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine.

Vol. XI | New York, December, 1913 | No. 12

PUBLISHED MONTHLY BY THE

McGraw Publishing Company, Inc.

James H. McGraw, President.
C. E. Whittlesey, Secretary and Treasurer.
239 West 39th Street, New York.
Telephone Call, 4700 Bryant. Cable Address, Metchem, New York.

Western Office.....421 Boston Bldg., Denver, Colo.
Chicago Office.....Old Colony Building
Philadelphia Office.....Real Estate Trust Building
London Office.....Hastings House, Norfolk St., Strand

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

J. M. Muir, Manager.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50. (European exchange, 10 shillings, 10 marks, 12.50 francs.)

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879.

New York, December, 1913.

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Organized Research in America

In his brilliant recent presidential address before the American Chemical Society, Mr. Arthur D. Little remarked pointedly that while "Germany has long been recognized as pre-eminently the country of organized research," yet its pre-eminence in industrial research is now being threatened by a new competitor, which is "strangely enough the United States—that prodigal among nations still justly stigmatized as the most wasteful, careless and improvident of all." While most of the industrial effort in this country still proceeds under the guidance of empiricism with a happy disregard of basic principles, yet many of our manufacturers have already learned that "efficiency of production is a sounder basis for prosperity than mere volume of product, however great" and that the most profitable output of their plant is "that resulting from the catalysis of raw materials by brains." Catalysis of raw materials by brains—this is industrial research.

Mr. Little's record of what industrial research in America has so far accomplished is a long and noble list of achievements, which promises much for the future. But what we want to comment on especially in this note is not so much what has been achieved as how it has been achieved—or as it has been sometimes put, though erroneously, "the question of individuality versus organization."

Most of the early achievements must be credited straight to the genius of strong individualities of superman size. Edison is a typical example. And that the Edison spirit is not yet dead, that specialized intense research is still being carried out by independent individualities and can be carried out with success, Dr. Baekeland has proved more than once. But it is true that under the new, more complicated social and industrial conditions of modern times most of the problems inviting research now require the facilities and resources of large corporations, and that organized research is replacing independent research.

Though this is perfectly true, it would be folly to imagine that organization can ever replace individuality. Research can be organized, but it will always require individualities of the most pronounced character. At this place last month we paid a tribute to the latest great achievement of the Research Laboraory of the General Electric Company—the two-candle per watt lamp. In Mr. Little's address we find an interesting account of how work is carried on at that laboratory. Dr. Whitney is quoted as follows: "We see a field where it seems as though experimental work ought to put us ahead. We believe that we need to get into the water to learn to swim, so we go in. We start back at the academic end as far as possible, and count on knowing what to do with what we find when we find it. Suppose that we surmise that, in general, combustible insulation material could be improved upon. We try to get some work started on an artificial mica. Maybe we try to synthesize

it and soon come to a purely theoretical question; e.g., is it possible to crystallize such stuff under pressure in equilibrium with vapor corresponding to the composition of real mica? This may lead a long way and call in a lot of pure chemistry and physical chemistry. Usually we just keep at it so that if you haven't seen it in the market we're probably at it yet." Clearly work of this kind requires strong individualities, but the work of the different research men is held together by their organization under a leading master spirit and by a mutual exchange of ideas between all at weekly colloquies.

Now the problem looms large: What can the Government, represented by the Government bureaus, do to help industrial research in this country? A broad, positive answer is difficult. But this much can be said that being a government of the people and by the people it should never undertake anything that would bring it into competition with individuals or classes of the people. The scope that remains is large. The Reichsanstalt in Germany has shown how to do it; so has the Bureau of Standards in Washington, which was justly congratulated at the last iron and steel meeting of the American Institute of Mining Engineers on its excellent research on the critical ranges A2 and A3 of pure iron. The problem is a peculiarly trying one for the new Bureau of Mines, but this Bureau is to be congratulated at least on its willingness to confer concerning the organization of its research work with the representative national scientific and engineering societies.

Competition Among Metallurgical Processes

Progress is rapid and change is frequent in the metallurgical industry. Just as one branch of ore treatment seems to have reached the limit of its efficiency, and the demand seems imminent for a radical change in methods, ideas are evolved that give the older methods a new lease of life and defer the possibility of success of the new proposals. We have in mind particularly the situation in regard to complex base-metal ores, and the relative possibilities of mechanical concentration, hydrometallurgy, and electrometallurgy in their treatment.

Advocates of each type of process are enthusiastic and optimistic as to the advantages of their respective favorites. Perhaps concentration is on the defensive; for the admitted limitations of physical processes have acted as a spur to the hydrometallurgist and electrometallurgist. At the same time, the deficiency of mechanical concentration, fully recognized by the loyal experts in that field, has forced them to perfect and adopt improvements that will strengthen their position against their possible competitors. And so the competition grows keener and keener, to the ultimate benefit of the industry.

There are several matters in this number of the journal that will afford food for reflection to all, and some degree of encouragement to each of these factions. The discussion on the present position and future prospects of zinc metallurgy at the recent joint meeting of the New York sections of the American Electrochemical Society and the American Institute of Mining Engineers, reported in full in this issue, is interesting in this respect, though the general discussion was rather tame compared with the discussion on the same subject at the last Denver meeting of the American Electrochemical Society. While the remarkable advances made in the concentration of complex zinc ores in recent years were not touched upon in the meeting at New York, yet the relative present position of the retort process, the electric-furnace process, and the electrolytic process came out very clearly, as well as the position of some of the chief representatives in the different camps, optimists and pessimists and agnostics.

Of great encouragement to the ore-dressing fraternity will be the note of improvement in dry concentration and separation, as recorded in this issue in the description and work of the Plumb pneumatic jig. It suggests possibilities that ought to be interesting to those who are not now getting satisfactory returns from their labors. It emphasizes the enhanced value of high-grade as compared with low-grade concentrates, and the necessity of working to meet the smelter's demands.

Imperfections reside in all of our processes, but the different types merit consideration. Their ultimate success or failure will be determined by the net profit resulting from their application. In this regard, a discussion started on another page, for and against a hydrometallurgical scheme, should serve a useful purpose. Several other communications referring to the same subject came too late to be printed in this issue and have been held over for the January issue. It is better to bring ideas to a focus in print than to talk aimlessly and in generalities. As suggested in the discussion, if concentration is to hold its own it must awake to the possbilities of improvement, and reduce the tremendous losses entailed in poorly separated products; and the goal of competitive metallurgical processes must be a cost of production so low that it can not be ignored.

The Modern Blast Furnace Burden

According to the official statistics the manufacture of 29,726,937 gross tons of pig iron in 1912 involved the consumption of about 55,656,000 gross tons of domestic and foreign iron ore, ore briquets, etc., and about 4,319,000 tons of mill cinder, scale, scrap, slag, zinc residuum, etc. As the latter item is only about one-thirteenth the former it does no great violence to add the two together, and compute that the average furnace burden was equivalent to 2.017 tons of ore per ton of pig iron manufactured, making a yield of a shade under 50 per cent. Without undertaking to go into details as to the loss of metallic iron in slag, flue dust, etc., it is evident that the average iron content of the ores used was less than 50 per cent, since the average pig iron contains only about 95 per cent iron.

Between 80 and 85 per cent of all the pig iron of the United States is made from Lake Superior ores, while the imported ores are of relatively high grade, leaving it that the comparatively low-grade ores used by Southern furnaces do not operate greatly to pull down the average yield. The showing is much less favorable than would have been made, say, fifteen years ago, when the cream of the Lake Superior deposits was being used. Unfortunately there are no statistics for such years, but it is readily recalled that ores running well above 60 per cent were the rule rather than the exception.

The ores have been changing and the conditions of operation change with them. The pig-iron producer adapts himself to the changes without inconvenience. The difference in conditions appears on the surface not in the form of its having become more difficult to operate a blast furnace, but from its being impossible to produce pig iron as cheaply as formerly. More ore must be mined and transported, and more fuel and flux must be consumed, simply adding to the expense.

It should be noted that the gradual exhaustion of the Lake Superior deposits which were first selected for exploitation does not always lead to the mining of a lower-grade ore. There are three factors, the iron content, the character of the impurities, and the cost of mining. In the early days of the Mesabi range, when ore was literally "as cheap as dirt"—certainly if one undertook to place orders for "dirt" in hundred-thousand-ton units he would find it rather expensive—if a 60 per cent ore lay in 100-ft. thickness on the surface and a 65 per cent ore of the same thickness lay under 100 ft. of overburden the former would have been mined and the latter left as impractical at the time, but to-day conditions would be different. Very good ores are now being mined, at great outlay of capital in removing great overburdens, and thus the exhaustion of the earlier mined deposits leads in one direction to the mining of lower-grade ores, but in another direction to the mining of deposits which are very good, but expensive to get at.

The ease with which pig iron is made with a much more expensive burden than that of fifteen years ago suggests how easy it will be to continue using lower and lower grade ores. We shall make tens of millions of tons of pig iron, perhaps nearly a hundred million tons, for every unit that the average ore used loses in iron content, and in making that pig iron we shall obtain ample experience to attack the new problems successfully. Inasmuch as the visible supply of iron ore increases very rapidly for each notch that the limit of commercial availability is dropped it is clear that iron for future generations can readily be found. A higher price, measured by unit of endeavor, will have to be paid, but through the endeavor being better directed the cost will not be enhanced in proportion. If the cost should advance when measured by dollars per ton, the public will undoubtedly have a correspondingly increased ability to pay. The consuming public can now pay the average price of $14 a ton which pig iron now commands much more easily than it could pay the $9 to $10 per ton which pig iron made from Lake Superior ores brought at its historic low point, late in the nineties, when very rich ores were being used.

An Old, Old Question Considered Prismatically and an Humanistic Hortation

Let us take a ray of sunlight and pass it through a prism. We all know what happens. The white light is decomposed into its optical components and we have the spectrum of red, orange, yellow, green, indigo, blue, and violet. What was a co-ordinate entity has been resolved into several entities and each not separate but blending into its neighbors.

Now compare the flux of social thought, which is made of many divers hues to a decomposed ray of light. We have two extreme classes of thinkers, one whose stomachs are full and whose souls are self-sufficient, and who quack solemnly "whatever is, is right" and we have another class of iconoclasts with lean and hungry looks, who say "whatever is, is wrong." Between these two opposed classes, we have those of lofty cosmocentric souls, who chant "whatever is true, must be right, let us advance slowly and surely onward and upward to the right." These classes grade into each other in many ways for the individual himself will be a conservative in one subject and a progressive in another subject. A man will yell "halt" and try to sit on the tail of progress in metallurgy while he will be a contributor to the funds of the Socialist party. We see that it takes all sorts of people to make up this round world.

Now all these discordant anthropological elements in the eyes of the Great Intelligence make up a white sunbeam of spiritual light. We should, therefore, be not too impatient with the "standpatter" if we have radical views on a question, nor damn with faint praise the efforts of the sane upbuilder if we are recognized by ourselves and by others as authorities on any subject. For physics teaches that a ray of light to be pure white must have many seemingly diverse elements.

Now physics teaches also that outside the visible spectrum, there is the invisible. Thus we have the infra-red which gives heat without light and the ultra-violet that gives light, could our purblind eyes detect it. So too, in politics and technology we have the commonplace reactionary who fires away, sputters and gives no real light on the points at issue and the far-sighted visionary who throws his rays into the projective and transcendental future and is described as a wild-eyed crank by those round about him.

But between our physical vision and our psychological vision there is this important difference. What colors our eyes can distinguish to-day we will see as the same to-morrow, next day, and forever. But what we see mentally as truth and reality does not necessarily seem the same to-morrow or next day, for we are being swept along in the great flow of thought and as the world moves, we nolens, volens must move with it. So what once was thought as unbelievable becomes belief, and the agnostic gets to know the unknown and sees the invisible.

This age is an intensely practical age. Such is a plain fact, but it is likewise a plain fact that this age is an intensively theoretical age. Most of the foregoing has been speculatively and theoretically considered. Now we will ask the reader to look at the question in a humanistic and direct-acting way, in a manner and in a way that will affect his own life. Which should a person be—a wise, cheerful, sane optimist or a crabbed, aged pessimist?

Now the sane optimist subdues his pessimism and uses it to cheer his spirit. He knows full well the motto "nisi paret, imperat."

Which should a right-thinking person be? Might he not consider this: "Between the optimist and the pessimist the difference sure is droll, the optimist sees the doughnut and the pessimist sees the hole." We may add that the calm, self-contained optimist usually eats the doughnut as soon as he can see it. And the traditional food of New England is more substantial than the circumspect hole or the circumambient air. The community of pessimist-subduing, early-rising optimists is a good society to join.

Readers' Views and Comments

Reduction of Blue Powder

To the Editor of Metallurgical and Chemical Engineering:

Sir: My attention has been called to an error appearing in my article in your November issue, page 637, in which I state, "Moreover, Dr. Haanel has informed me that he has melted tons of blue powder, after mixing it with carbon in proper form and amount." Although I believed I was stating a correct impression received in conversation, it now appears that the statement is unqualifiedly wrong, and I wish to make a complete retraction to relieve Dr. Haanel from any wrong implication.

We had agreed that the excessive production of blue powder was not necessarily objectionable if it could be cheaply converted into spelter. Probably it was in this connection that I misunderstood Dr. Haanel's remark.

Denver, Col. F. L. Clerc.

Electric Tin Smelting

To the Editor of Metallurgical and Chemical Engineering:

Sir:—On page 593 of your October issue an article is referred to under the heading "Electric Tin Smelting," given by "A Special Correspondent" in the *South African Mining Journal*, June 7 and 14, 1913.

On reading this through I find that practically all the information given therein is identical with the data and conclusion given in my article, "Electric Tin Smelting," published in your journal, Vol. IX, No. 9, page 453, September, 1911, although no reference to that article is made in the review.

Luton, Beds, England. Joh. Härdén.

Annual Tables of Physical and Chemical Constants

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I desire to make public acknowledgment of contributions made this month toward the expenses of publishing the "Annual Tables of Physical and Chemical Constants": New Jersey Zinc Company, $50; International Acheson Graphite Company, $25; Castner Electrolytic Alkali Company, $25; Mathieson Alkali Company, $25.

Lehigh University. Jos. W. Richards.

Induction Furnace Notes

To the Editor of Metallurgical and Chemical Engineering.

Sir:—In my article on the above subject, published in the October issue of your journal, a few minor errors occurred, which I now wish to correct.

(1) As to the thermal efficiency (page 561) an error appeared in the formula given. It should read

$$E = \frac{350{,}000}{864 \times 606} = 67.1 \text{ per cent, not } 63.8 \text{ per cent.}$$

(2) The cooling water was found to be just under 1.4 gal. per min. or round 50,000 gal. for 160 tons of steel. As the cost of the water from the mains used is 6d. per 1000 gal. This works out to a little under 2d. per ton of steel.

(3) The tilting arrangement is not used to any extent at present, as it was found that the use of a sound taphole was more beneficial to the lining generally, and also probably to the quality of the ingots.

(4) The tar-magnesite lining has again been abandoned. The first mentioned method of lining has been taken into regular use, though with a few technical alterations, the nature of which is not divulged for publication. When the men got thoroughly used to its behavior, cracks in the lining are now a very rare occurrence and need not be feared.

(5) The labor casts have been revised and are slightly higher than given in my schedule; in addition, two "shank men" at 8s. are employed per shift. The smelters receive about 9s. each per shift, and the laborers' wages are also slightly higher. Against this stands the fact that the necessary margin for sundries has been materially lowered so that the total cost per ton remains pretty much the same as stated.

Luton, Beds, England. Joh. Härdén.

Micrometry as Applied to Alloys

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The review of the very interesting paper, "Micrometry as Applied to Alloys," by Dr. C. H. Mathewson, in your November issue, calls to my attention a method of measuring the areas of alloy constituents which I have used very successfully. It is a modification of Sauveur's polar planimeter method combining its accuracy with the shortness of the ruled square method.

Two ways of measuring areas with the planimeter are now in general use: (1) taking a photomicrograph of the alloy and measuring the areas from a print, and (2) projecting an image of the alloy on a screen and tracing the areas to be measured with a lead-pencil. The penciled areas are then measured with a planimeter.

I used a screen consisting of a piece of plate glass, ¼x15x15 in., coated on one side about 1/32 in. thick, with a mixture of paraffin and tallow If now the coated glass be used with the waxed side up, in conjunction with a Bausch and Lomb photomicrographic camera, the areas can be traced in the wax with a planimeter according to Sauveur's method. The camera should be in a vertical position thus making the glass horizontal. It is best to confine the waxed portion of the glass to an area about 4 in. diam. In this way the fixed end of the planimeter can be so placed that the roller moves on the glass without danger of encountering the wax.

The wax used was obtained from candles and the amounts of paraffin and tallow were not determined. It is evident that an opaque wax is desirable since it permits the use of a thinner coating. If the coating is too thick and the areas to be measured are small, the tracing point of the planimeter has a tendency to break the wax when an area is about seven-eighths surrounded. In discussing this difficulty with Mr. F. A. Fahrenwald, 2168 Stearns Rd., Cleveland, O., he suggested mixing some white powder with paraffin to make it more opaque and thus permit the use of a thinner layer. Of several powders tried we found that starch was the best. The paraffin is first melted and then powdered starch added slowly with vigorous stirring, until the mixture is pasty. We found that glass coated with a layer of this material .008 in. thick would give a much better image than ground glass.

After a determination has been made the wax can be melted by applying the flame of an inverted bunsen burner, after which the plate is ready for another determination.

Zay Jeffries.

Case School of Applied Science, Cleveland, Ohio.

Electric Zinc Smelting

To the Editor of Metallurgical and Chemical Engineering:

Sir: Referring to the article on Electric-Zinc Smelting by Mr. Louvrier in your November number, will you kindly allow me space to disclaim the intention of imputing to him, in my paper of last August, the expression of the opinion that "the production of carbon dioxide in actual electric furnaces, was approximately equal to that formed in the Belgian retort."

On the contrary, I understood him to say that "the excessive production of blue powder in electric furnaces is due to the formation in the electric furnace of a much higher percentage of carbon dioxide than in the usual retort."

I expressed as my own opinion, that either he was mistaken in this opinion, or that the excess of oxygen came from another source; for the reaction $ZnO + CO = Zn + CO_2$ will give the same amount of CO_2 in either case.

I also ventured to express my opinion that the reaction $CO_2 + C = 2CO$ would be equally rapid in both cases.

I also stated that in my opinion, the electric current, by forming cavities in a pasty, partly fused charge, might prevent the intimate contact required between the carbon dioxide formed and the hot carbon, in order that the second reaction could take place.

I offered as proof of this, that a rapid reduction of zinc could not take place unless the carbon dioxide was rapidly reduced, and suggested that some of the carbon dioxide escaping into the slag-lined passageways, might pass into the condenser without an intimate contact with carbon.

The amount of carbon dioxide formed being the same in both cases, the percentage of it which passes into the condenser in an electric furnace, may thus be much greater than in the Belgian retort. This may be the result either of a defective construction in the particular furnace under consideration, or of some inherent difficulty in reducing zinc at the temperature of an electric furnace.

At the time my article was written I had no idea that Mr. Louvrier had designed a zinc furnace, and, of course, had no idea of criticising it.

This is plainly a case of misunderstanding which should be avoided if possible, and I wish to dispel in the minds of Mr. Louvrier, and any of your readers, the thought that I have treated him unfairly.

Denver, Col. F. L. CLERC.

Hydrometallurgy: Joys of Its Theory, Woes of Its Practice

To the Editor of Metallurgical & Chemical Engineering:

Sir: In reading Dr. Chauvenet's objections to "hydro schemes," in your September issue, I wondered why he selected for attack dry chlorination as typical of hydrometallurgical processes, for not a single "woe" that he gloats over applies to the method in its present development. He surely must have had in mind chloridization of a period much earlier than the "early attempt at Helena," or the developments at Corbin, Mont. As well take as an example of current cyanide practice, the process in its stage of development fifteen years ago, as to refer to chloridization today in terms of concentration roasting, sodium chloride electrolysis, ferric hydroxide precipitation, water washes, diluted solutions and fractional precipitation.

The prominence of dry chlorination as the most likely process in the field for the solution of the complex-ore problem is attested, not only by a score of engineers of unquestioned ability, but now by its unexpected selection as a typical example of "hydro schemes," with no specific "woe" charged against it.

If overcoming a simple problem of eliminating ferric hydroxide is to be termed "much ingenuity," "we" would bankrupt the English language in trying to express what has been done to overcome the *real* difficulties which have been encountered and successfully solved in the development of chloridization.

What engineer of real training and experience would offer as proof of the success of any hydrometallurgical process, the results of a test, whether on a gram, pound, ton or carload, if such test had not been made under commercial conditions of apparatus, reagents and time. Continuity of operation has no small bearing on the testing of a process. I do agree that engineers should know the relation of test work to actual practice. My experience has convinced me that the principal difference between a test on a few pounds and one on a ton (if both are intermittent) is the difference in the cost of making such tests.

Why eliminate as a class of material for wet treatment the available complex zinc ores which are not too low-grade, but rather too complex? I submit that in the consideration of any scheme, its field of operation and the purpose for which it is being developed should determine largely its possibilities. In other words: Will we "get a run for our money?"

In the October issue of your journal, Mr. Bleecker defends in a general way hydrometallurgical possibilities; but he has touched upon chloridization under the so-called Malm process just enough to expose his ignorance of the subject by his reference to its theory and calcium sulphate.

I am interested in the development of chloridization solely from an established conviction that it offers the solution of the complex-ore problem. To this end I am prepared to defend it, and welcome any sincere, specific criticism devoid of repartee or hair-splitting. This will offer to Mr. Bleecker an opportunity to enlighten us further on "calcium sulphate in the Malm process," and I trust will make available these columns for a discussion of the relative merits of various proposed schemes of dry chlorination.

To establish my position, I state that my experience with chloridization has been through the gram to the ton test in as thorough an investigation as intermittent tests will permit; then to a test with commercial-size apparatus under conditions indicated by the work on a smaller scale. This latter installation was made at Corbin, Mont., and proved a failure under the methods which the preliminary tests indicated would work successfully. From this failure was evolved a method which of necessity discarded all specially constructed apparatus, and made use of standard apparatus excepting only the special construction of minor parts and the development of a simple, cheap, fool-proof electrolytic cell for the decomposition of zinc chloride.

This work covered a period of three years, during which time the weak points of the process became apparent and were overcome. The difficulties encountered were solved in such a way as to simplify the method, so that in its present development all of the "bugaboos" mentioned by Dr. Chauvenet (many of which never existed) have vanished into thin air.

I stand for a form of chloridization which, at least to myself, has proved a success in yielding a high recovery of gold, silver, copper, lead and zinc in metallic condition at a cost, under normal Colorado conditions, of 2½ cents per pound of metal recovered. In the last analysis the success of any commercial operation is determined by its net financial return.

I therefore have no proof of the success of chloridization. If such were the case the plant at Georgetown, Colo., would not be in its uncompleted state. It was to bring the process to a stage of commercial proof that the installation at Georgetown was undertaken. At present the plant is about two-thirds completed. It was designed on data obtained from the Corbin plant and from later special investigations, to treat all ores, whether sulphide, oxide or carbonate, but more particularly complex ores of gold, silver, copper, lead and zinc. The present installation bears evidence as to the comparative simplicity of apparatus and operation proposed.

During the eight years which I have given to this method of ore treatment, I have not encountered a difficulty which has not been readily solved, and no difficulty is foreseen which is any greater than those which are the regular diet of engineers who do things. Oh, yes, "the cost will finally prove its death!" In that connection, let me state that the value of ores available will permit the cost to more than double and still show a substantial profit on ores which have no market value at present.

JOHN L. MALM.

Denver, Colo.

Great Falls Electrolytic Copper Refinery

To the Editor of Metallurgical and Chemical Engineering:

SIR:—The paper on the "Great Falls Electrolytic Plant," by Willis T. Burns, presented at the Montana meeting of the American Institute of Mining Engineers and published, slightly abstracted, in your September issue, has brought up some thoughts which I herewith offer.

Re Purification of Electrolyte by means of insoluble anodes, the writer states that using insoluble anodes with a speed of flow of electrolyte of 4 liters per minute four tanks removed 99.3 per cent of the arsenic, leaving the bismuth still in solution. In December, 1910, Harold Schroeder in the *Australian Mining Standard* published details of a similar method of purification of electrolyte conducted at Lithgow, N. S. W., in 1908.

The object he concisely states under "*Scheme*" as "to purify the liquor and at the same time produce the least possible quantity of impure copper." To effect this, as much pure copper as possible is deposited by sending the liquor through a cascade of tanks with insoluable anodes just sufficient to ensure the lowest one giving pure copper on last cathode. In this plant five tanks were used, for the double reason that that number necessitated no alteration in the ordinary flow arrangements and that it still left the ordinary number of insoluble-anode tanks in use for keeping the copper strength of all electrolyte on works at a constant figure. The outlet from the fifth tank was used to fill tanks in which the impure copper was to be made, no circulation of liquor taking place in those tanks beyond the ordinary air-lift circulation in each tank.

The figures published show a steady and regular daily deposit of the arsenic, but the bismuth was deposited much more rapidly than the arsenic. In fact where seven days were required to remove the bulk of the arsenic, the bismuth was all removed from solution in less than half that time. This is confirmed by the analysis of the deposits, these being divided into hard cathode, metallics and fines of the powdery part, the ratio of bismuth to arsenic always being higher in the metallic portion.

Having personally supervised the whole of this experimental purification, I may say that what suggested this method of cleaning electrolyte was largely, having noted on several occasions that when the circulation of liquor was accidentally stopped or slowed down during the night in the insoluble anode tanks, the tanks at the end of the cascade produced dirty copper, and sometimes a cathode had the top 6 inches black and the lower part clean bright copper. Samples of electrolyte from the tank containing such a cathode, drawn off in zones from top to bottom of tank, showed differences as to copper, arsenic and bismuth contents. To this H. Schroeder draws attention.

Our experience on the Australian works differs completely from that given in Mr. Burns' paper.

It would be of interest to know if the removal of the first of the copper by crystallization has been found more economical than the making of pure cathode copper till the solution reached the "danger point" for such.

Re the Treatment of the Slimes and Impure Copper Produced, it would also be of interest to know whether any other method than blast-furnace treatment has been tried; further, whether the anodes cast from converters at a time when much of this impure copper was coming around, carried a higher percentage of arsenic than the average anode. It is worthy of time to evolve a method of treatment by which a minimum of the arsenic would eventually return to the liquor, whether from blast-furnace matte or the after-treatment of flue dust from the roasters or converters that expelled it from the impure copper deposit.

In another part of the paper the writer states, "all of the *silver present in the cathodes* is deposited mechanically from the slimes in suspension." This I find very interesting as bearing out what I have always maintained, and we have proved conclusively at the works at Lithgow by the use of sand filters and by analysis of cathodes. Details are given in E. H. Blakemore's paper on "Electrolytic Refining," read before the Australian Institute of Mining Engineers, and published in its *Proceedings*, February, 1910.

It is to be regretted that the author of the paper on the Great Falls plant makes no comment on the use of chlorine in the electrolyte, which I see was kept at 0.40* grm. per liter. In Lithgow we proved it to have a decidedly bad effect in relation to the gold and silver in the cathode copper.

Gourock, Scotland. HARRY A. B. MOTHERWELL.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Replying to Mr. Harry A. B. Motherwell's comments above on my notes on the Great Falls Electrolytic Plant reprinted in your September issue I wish to say that the bismuth in the copper treated at Great Falls is present in very small quantities, as compared with the other impurities, and as it has shown no tendency to increase in the electrolyte its elimination has not been studied.

Since the receipt of Mr. Motherwell's comments a bismuth determination has been made on samples of the inlet and outlet solutions of a group of four insoluble anode tanks employed in purifying electrolyte. No bismuth was found in either sample. Had bismuth been present in these solutions it would probably have been deposited with the arsenic.

Regarding the economy of first removing the copper from the impure electrolyte, by crystallization, before treating in the insoluble anode tanks for the removal of impurities I may say that this method has been found much more economical than attempting to obtain commercially pure copper in insoluble anode tanks operated at 35 amperes per square foot when the electrolyte is circulated at a rate of speed sufficiently slow to insure the deposition of a large amount of arsenic. Even were it possible to recover most of the copper in this way we would find the scheme impracticable as our working electrolytes would become depleted in copper as the increase in copper contents of the electrolyte, per ton of copper refined, decreases as the current density increases.

Regarding the arsenic contents of anodes produced at the smelter while the arsenical slime from the insoluble anode tanks is being treated there, it may be said that no material increase in impurities in the anodes is noted.

The purpose of adding chlorine to the electrolyte is to precipitate antimony in the form of an oxychloride. We have observed no effect of the chlorine on the silver and gold contents of the cathodes.

W. T. BURNS.

Great Falls, Montana.

*This figure should be 0.044. The figure in the September issue was a misprint.—EDITOR.

Conductivity of Fluosilicic Acid and Lead Fluosilicate

To the Editor of Metallurgical and Chemical Engineering:

Sir: As there is, to my knowledge, no published data on the conductivities of fluosilicic acid and lead fluosilicate, the constituents of the electrolyte used in electrolytic lead refining, I thought the following tables might be of interest.

These results were obtained in the Electrochemical Laboratories of the Pennsylvania State College. The method employed was that familiar to anyone who has worked with conductivities, namely, the one due to Kohlrausch, which is essentially the standard Wheatstone bridge method of resistance measurement, with the exception that an alternating current is substituted for a battery and a telephone receiver for a galvanometer. Before making any measurements the slide-wire bridge was calibrated by the method of Stronhal and Barus, and all subsequent readings on the bridge were corrected from the calibration curve obtained. Two conductivity cells, having platinized-platinum electrodes, were used, one cell having a high and the other a low "cell constant."

The acid used in making up the solutions was the pure concentrated fluosilicic acid, a clear syrupy liquid, bearing the following analysis:

N. V. M. = 0.095 per cent

SO_3 = 0.002 per cent

The salt used was the crystallized lead fluosilicate of the formula $Pb\,Si\,F_6\,(4\,H_2O)$.

All results given in the tables are the means of at least three readings.

Table I gives the conductivity and equivalent conductivity of fluosilicic acid at 25 deg. C.

TABLE I

Conductivity and equivalent conductivity of fluosilicic acid at 25 deg. C.

Normality	Per cent of acid in solution	Conductivity in reciprocal ohms per centimeter cubed	Equivalent conductivity
N	7.2	0.087	87.
N/2	3.6	0.0469	93.9
N/4	1.8	0.025	100.
N/8	0.9	0.0133	106.5
N/16	0.45	0.00708	113.2
N/32	0.23	0.00372	119.9
N/64	0.11	0.00192	123.
N/128	0.06	0.00102	130.9
N/256	0.03	0.000602	154.
N/512	0.015	0.000384	196.9
N/1024	0.007	0.000254	271.
N/2048	0.0035	0.000156	320.

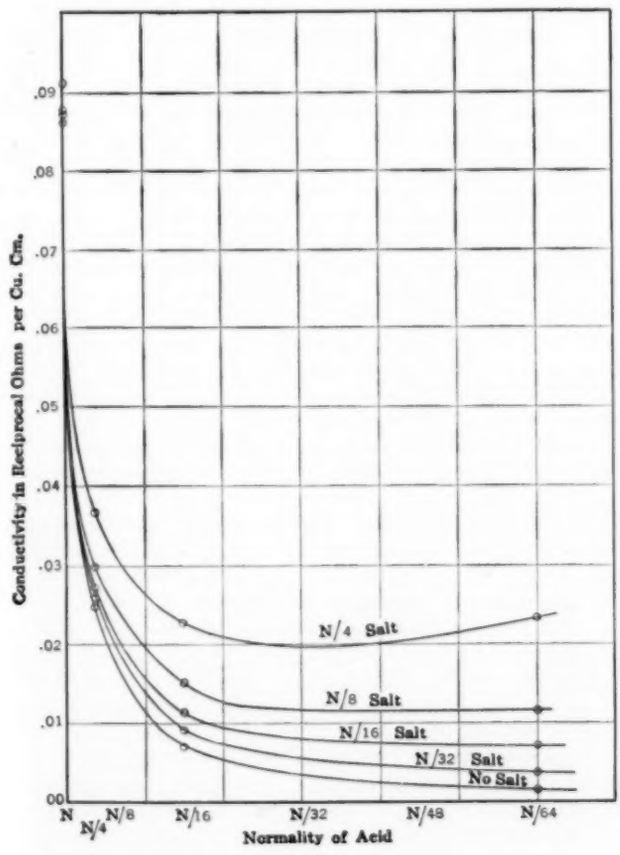


FIG. 1.—EFFECT ON THE CONDUCTIVITY OF FLUOSILICIC ACID OF DIFFERENT NORMALITIES OF LEAD FLUOSILICATE DISSOLVED IN THE ACID

TABLE II

Temperature coefficient of conductivity of normal fluosilicic acid (corresponding to 7.2 per cent free acid) from 20 to 45 deg. C.

Temperature in degrees centigrade	Conductivity in reciprocal ohms per centimeter cubed	Equivalent conductivity
20	0.0806	80.6
25	0.0866	86.6
30	0.093	93.
35	0.099	99.
40	0.104	104.
45	0.1092	109.2

Temperature coefficient of conductivity equals 0.01459 per degree centigrade between 20 and 45 deg. C. referred to 18 deg. C.

TABLE III

Temperature coefficient of conductivity of 1/16 normal fluosilicic acid (corresponding to 0.45 per cent free acid) from 20 to 45 deg. C.

Temperature in degrees centigrade	Conductivity in reciprocal ohms per centimeter cubed	Equivalent conductivity
20	0.00669	107.
25	0.00715	114.2
30	0.00754	120.6
35	0.00792	126.9
40	0.00820	131.1
45	0.00849	135.9

Temperature coefficient of conductivity equals 0.001106 per degree centigrade between 20 and 45 deg. C. referred to 18 deg. C.

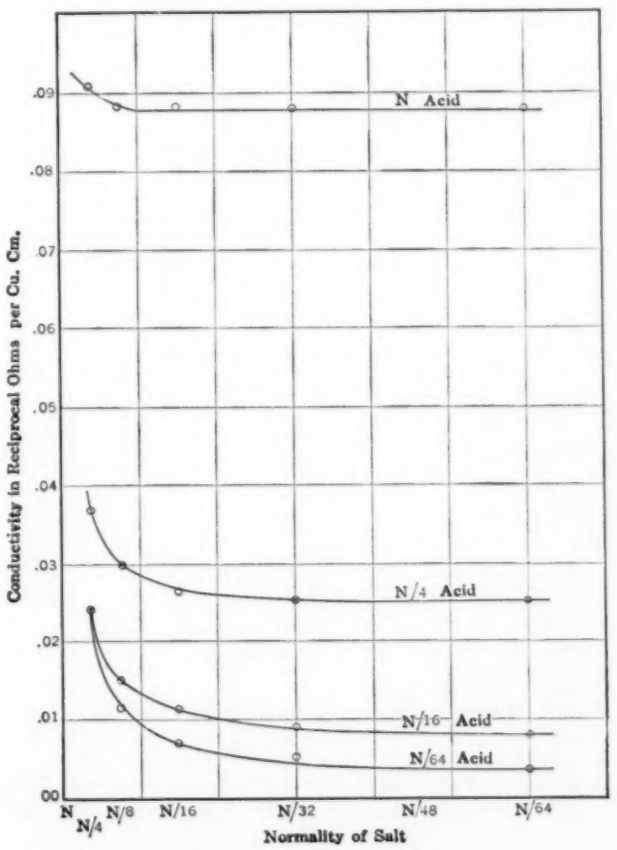


FIG. 2.—EFFECT ON THE CONDUCTIVITY OF LEAD FLUOSILICATE WHEN DISSOLVED IN DIFFERENT NORMALITIES OF FLUOSILICIC ACID

TABLE IV

Conductivity table of various mixtures of fluosilicic acid and lead fluosilicate at 25 deg. C.

	N acid	N/4 acid	N/16 acid	N/64 acid
N/4 salt	0.0908	0.0367	0.02278	0.0230
N/8 "	0.0884	0.0300	0.01519	0.01175
N/16 "	0.0884	0.0273	0.01122	0.00725
N/32 "	0.0876	0.0258	0.00932	0.00470
N/64 "	0.0876	0.0254	0.00838	0.00362
N/128 "	0.0876	.....	0.00792	0.00265
N/256 "	.....	.....	0.00772	0.00224
N/512 "	.....	.....	0.00759	0.00206
N/1024 "	.....	.....	0.00750	.....
No salt	0.0870	0.0250	0.00708	0.00192

Table II gives the temperature coefficient of the conductivity of normal fluosilicic acid (corresponding to 7.2 per cent free acid) from 20 to 45 deg. C.

Table III gives the temperature coefficient of the conductivity of 1/16 normal fluosilicic acid (corresponding to 0.45 per cent free acid) from 20 to 45 deg. C.

Table IV is a conductivity table of various mixtures of fluosilicic acid and lead fluosilicate at 25 deg. C.

When the results obtained in Table IV are plotted, as in Figs. 1 and 2, they show, at a glance, the great effect that the free acid has in increasing the conductivity of the electrolyte.

Niagara Falls, N. Y. T. Arthur Patterson.

The Continuous Sand Plant in Cyanide Practice on the Rand

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In your issue of July 1913 appears an article by Mr. H. N. Spicer in regard to Rand methods, and under the heading "Cone Classification of Sand and Slime" Mr. Spicer refers to rotary vacuum filter tables of local invention in use at the City & Suburban G. M. Co., and alleges certain disadvantages in connection with the use of these tables.

The tables referred to, and also the special type of cone used in conjunction with same, are the invention of Dr. W. A. Caldecott, and the method of handling sand by means of table and cone in the manner described is known as the "Caldecott continuous sand plant," and as the writer has had a great deal to do with the installation of these plants, as Dr. Caldecott's representative, he would like to comment upon Mr. Spicer's article. The alleged disadvantage, as stated, are based on faulty premises.

Loss of Head.—Mr. Spicer seemingly overlooks the fact 85 per cent of the total weight of pulp, in the shape of slime and water, overflows the diaphragm cones, and return-sand cones above the filter table, and the total loss of head of this amount of pulp is only about 2 ft., which can hardly be considered a serious matter. The remaining 15 per cent of the pulp is the only part which has any considerable loss of head, as it is this amount of the total pulp which is delivered as a thick candy pulp from the underflow of cones on to the sand filter table.

These percentages are based on ordinary Rand conditions of 5½ parts water to one of solids in the pulp from the mill, and with cone underflow containing not more than 30 per cent of moisture. The sand delivered by underflow of cone amounts to 50 per cent of the original solids; the other 50 per cent being slime which overflows cones.

The disinclination to any loss of head requiring elevation and return of a portion of the pulp in a reduction works, whatever advantages may be derived therefrom, and however advantageous it may be found elsewhere, is quite a marked feature of the older American and Mexican ore-treatment practice, and is, apparently, due to the fact that, in the early days, the side of the hill was always selected as the site for the reduction plant, and full advantage was invariably taken of the use of gravity for transferring pulp from one stage of treatment to another, and any departure from the gravity system has been adopted very slowly and cautiously, but in the more modern American and Mexican plants there is a growing tendency to the use of circuits in milling, especially in tube-milling, and the use of a circuit necessarily means the re-elevation of some part of the pulp. In connection with this matter the article by Herbert Lang in the July 5 number of the *Mining & Scientific Press*, page 7 is of interest as showing the trend of modern metallurgical practice.

In regard to the double treatment mentioned by Mr. Spicer, I presume he refers to the use of return sand cones, but, as stated above, the use of these cones only involves the loss of about 2 ft. of head on 85 per cent of the total pulp.

Sand Filter Tables.—The cost of operating Caldecott Tables at the City & Suburban G. M. Co., referred to in Mr. Spicer's article, expressed in American currency, is as follows:

	Per Ton of Sand:	Per Ton of Ore:
Power	2.36 Cents.	1.31 Cents.
Maintenance	0.43 "	0.23 "
Labor	0.58 "	0.33 "
Totals	3.37 Cents.	1.87 Cents.

The value of the original sand is 3.52-dwt., and the value of the residue is 0.371-dwt.

It will be seen from the above figures that the metallurgical efficiency of the system is high, and that the cost of power and maintenance is very small, which is due, principally, to the fact that the table revolves at a very slow speed, viz: one revolution in 3 minutes. In regard to the cleaning of filter cloths, this work is performed in half-an-hour by two natives.

Necessity for Two Tables.—While in all continuous operations it is desirable to be able to cut out a unit periodically still it has been found possible, on the Princess Estate & G. M. Co., to work with one filter table and yet obtain most satisfactory results under proper supervision as can be seen by reference to Mr. H. A. White's contribution to the discussion on the Metallurgy of the Homestake Ore, which appears on p. 55 of *Bulletin* No. 100 of the Institution of Mining & Metallurgy.

In Mr. White's article he shows that the sand residue at the Princess Estate, over a period of three months, had an average value of 0.23-dwt., so it can be seen that the use of one table only gave excellent results. Mr. White's article also shows the extremely low value of last drainage solution of 0.05-dwt., and this is due to the fact that, by the Caldecott system, the slime is completely eliminated, resulting in very clean sand.

At the Princess Estate the sand treatment is what is termed "single" treatment, i. e., the extraction of gold from the sand is completed in the vat into which the sand is transferred from the filter table, and the residues are so low that they would not pay for transferring into another vat for the purposes of being given a further treatment.

The advantages claimed for the Caldecott patent continuous sand plant are as follows:

I. Efficient classification of sand and slime in tailing pulp.
II. High percentage of slime which can be treated by cyanide at less cost than sand.
III. Clean, leachable sand charges, permitting free aeration, and facilitating the dissolving of gold, and the washing out of gold with decreased volume of solution.
IV. Clean sand residue, for sand filling.
V. Low cost of plant operation and maintenance.
VI. Lower capital expenditure for plant.
VII. Dissolving of gold in sand by cyanide solution begins within half-an-hour of ore being crushed in battery.
VIII. Less volume of gold-bearing solution to precipitate.

The validity of the foregoing claims is evidenced by the fact that at present about 20 per cent of the total Rand tonnage is being treated by this process, and the following is a list of companies using the Caldecott process, together with the respective ore tonnages treated *per month*.

	Tons
(a) Simmer & Jack Proprietary Mines, Ltd.	79,700
(b) Jupiter G. M. Co., Ltd.	42,400
(c) Simmer Deep, Ltd.	63,000
(d) Princess Estate & G. M. Co., Ltd.	22,800
(e) Geduld Proprietary Mines, Ltd.	24,000
(f) East Rand Proprietary Mines, Ltd.	170,000
(g) New Kleinfontein G. M. Co., Ltd.	45,000
(h) City & Suburban G. M. Co., Ltd.	27,500
Total	474,400

Crushing in Water.—Mr. Spicer writes as though there is nothing in favor of the Rand practice of crushing in water, and while impartial judges readily admit the desirability of crushing ore in cyanide solution where the percentage of recovery by amalgamation is trivial—as in connection with gold telluride, or silver sulphide ores—still,, they are equally ready to admit the fallacy of crushing in cyanide solution where a high percentage of the gold can be recovered by amalgamation.

It is certainly unsound, metallurgically, to crush Rand ore in cyanide solution and thus needlessly incur trouble from corroded copper-plates, liability of loss of gold-bearing solution, base bullion due to deposition of dissolved copper and zinc, and the more prolonged and expensive cyanide treatment necessitated by richer tailings. Further, crushing in cyanide solution renders it impossible to obtain accurate screen values, due to the presence of dissolved gold in solution.

The crushing of Rand ore in cyanide solution has been tried on a large scale, and abandoned for the reason stated above, and as a great deal of expense was incurred in converting mills to make them suitable for crushing in cyanide solution, there was, of course, every inducement not to revert to crushing in water, but the troubles experienced were so great that such a change was found to be absolutely necessary.

In America you have the Homestake Co., treating an ore of about the same characteristics as regards amalgamation as Rand ore, and it is interesting to note that the Homestake crushes in water.

Rand Metallurgists are becoming rather used to outside criticism, and an article appearing in the August 9th, 1913 issue of the *South African Mining Journal*, under the heading "Rand Metallurgical Methods," would doubtless interest some of your readers. It reads as follows:

"In two or three recent issues of this journal we have printed extracts from overseas critics of Rand mining and metallurgical methods. The views of gentlemen like Mr. Motherwell, of Australia, and Mr. Spicer, of America, the two other large gold producing continents of the world, have, we understand, attracted much local attention, and it is generally felt that their comments on Rand metallurgical methods are scarcely just.

"Briefly stated, most of these overseas critics, while applauding the general organization and magnitude of operations on the Witwatersrand, find fault with the metallurgical practice of the Main Reef series. One or two of them attack the methods pursued in mills and cyanide works on the ground that they are antiquated and conservative. Other technical visitors who have recorded their impressions appear to find fault with Witwatersrand metallurgists in that they have borrowed ideas from all other mining countries.

"On the face of it these statements seem contradictory to one another. If the argument that our equipments and devices are obsolete is tenable, then the view that the Rand is to be condemned because our technical experts have not been slow to appreciate the value of the ideas of other mining camps cannot very well be supported, for it obviously infers a denial of the first-named statement.

"The essential view which most of these critics seem to lose sight of is that the primary object of mining is the earning of profits for shareholders. The enunciation of this basic economic fact leads one on to a consideration of a number of logical corollaries. In the first place it should be recognised that progress in ore treatment means making more profits from the same grade of ore. It may or may not, however, involve (a) new processes or devices—a 'different' method or device is not necessarily a 'better' one; (b) a reduction in working cost; (c) higher percentage recovery.

"Critics who have machinery or processes to exploit consider that the employment of a new device or a new method is synonymous with progress. Often they do not take best results but those which favor their own purposes. Progress may involve the use of new devices or the employment of new methods, or it may consist in (a) better use of existing appliances; (b) economy through better arrangement of known devices; or (c) better organization of work.

"On the Rand all types of practice are represented, from those of the pre-war epoch to those of 1913, but no individual existing plant represents all known 'best' practice. During the last few years there has been a large accumulation of knowledge requiring to be digested and profitably applied. Local conditions make many devices and methods unsuitable here, as for instance, the large scale of operations precludes the 'small bandy machine,' of limited capacity, just as a 3s. per day Kaffir may be more economic than a mechanical device with an 8s. per day white man.

"Another important consideration is that many devices have been tried locally and have been found wanting, which are claimed to work well elsewhere. The results of the successes only are made public; experience with failures is generally kept quiet for obvious reasons. Then, too, it must not be overlooked that the low average grade of ore and large sums of capital in plants and interests involved on the Rand preclude rash changes being recommended by those who are trustees for shareholders, and gambling merely in the name of progress with shareholders' money is a breach of faith.

"Notwithstanding all this, it must be admitted by all unprejudiced critics that Rand practice is good. At present banket is treated with a total residue at 0.3 dwt. at a total cost from shaft to dump of 3s. 2d. per ton. In fact, Rand practice of today represents combined efforts of many able men from every part of the world for a quarter of a century; the combined result of large capital and experimental expenditure. It is essentially a process of evolution and survival of the fittest. Note, too, that there is very little scrapped machinery, representing waste of shareholders' money on these fields.

"Gradual development has usually admitted of utilization of existing plant combined with its extension on more modern lines.

"Another important aspect of progress is this: At any time and on any mine there is an 'economic limit' of extraction or residue beyond which it does not pay to go, since such extra gold costs more than it is worth. Other essential facts to remember are the simple nature of the Rand ore which does not require many appliances in use elsewhere. The large scale of operations makes organization of great importance, and the group system with specialist staff at the disposal of each mine gives these mines advantages single companies cannot afford.

"To sum up, we believe that in crushing practice, in stamp and tube milling and classification with limited supply of water the Rand is ahead of the rest of the world. When an appliance from abroad (*e. g.*, tube mill) which is suited for use here arrives it is promptly adopted and fully utilized. There are probably more tube mills in use on the Rand today than anywhere else in the world."

It occurred to me the article quoted may interest your readers as representing the Rand side of the question.

EDW. L. BATEMAN

Johannesburg, South Africa.

The Western Metallurgical Field

National Radium Institute

For over a decade the carnotite deposits of southwestern Colorado and southeastern Utah have been exploited in a desultory way. They have been regarded mainly as possible sources of vanadium and uranium, and up to two or three years ago the ores were mined and treated for those metals exclusively. Recently, however, a new phase has been put on the whole situation by the general recognition of these ores as a source of radium; and the demand for salts of the latter element has lent an added value to the deposits. Not only has this district come into prominence in the eyes of American scientists, but it has gained a world wide reputation, so that foreign capital has sought to purchase the ores mined there, or failing in that, to buy mining claims.

The vast change in the situation has been caused by a realization of the radium value of these ores; and with that realization came the feeling on the part of some of our altruistic scientists that this important resource should be conserved for the benefit of our own country; that the radium-bearing ores should be treated here instead of in Europe, and the radium placed at the disposal of American scientists. The result is the organization of the National Radium Institute, directed by Dr. James Douglas, of Phelps Dodge & Co., New York; Dr. Howard A. Kelley, of Johns Hopkins University, Baltimore;

Dr. C. F. Burnam, associate of Dr. Kelley; Mr. Archibald Douglas, nephew of James Douglas, and Mr. E. J. Maloney, of Wilmington, Del. The institute has provided capital for the purchase of carnotite claims, and the mining and treatment of the ore. A cooperative agreement exists between the institute and the national Bureau of Mines, whereby the latter carries on the technologic study of the ore treatment and supervises for a time the actual process of recovering the three valuable constituents, uranium, vanadium and radium. The institute is a philanthropic body to the extent that its capital is to be expended in the production of radium which will not be sold, but used for scientific purposes.

The radium value of American uranium ores can be best comprehended from the following facts: In 1912 the United States produced 28.8 tons of U_3O_8, equivalent to 24.4 tons of metallic uranium. All but a few tons of this was shipped abroad. On the basis of the normal equilibrium between radium and uranium, this production represented 9.77 grams of radium chloride or 12.7 grams anhydrous radium bromide. But on the basis that the ratio of radium to uranium in carnotite is about 10 per cent below the equilibrium value, the above uranium production in 1912 contained radium equivalent to 8.8 grams radium chloride or 11.43 grams radium bromide. Contrasted with these figures we have the fact that the Austrian production of radium chloride in 1912 was only 2.23 grams. In making this comparison it must be remembered that the figures from Austria represent radium chloride actually produced, while those for the United States give only the radium content of ore shipped from the country; but allowing a recovery of 85 per cent of the radium in the United States ore, it is seen that our local production was still three times as great as that of Austria, the principal producer.

The reduction plant of the National Radium Institute will be built in Denver, where it can be under the immediate supervision of the officials of the Bureau of Mines who are charged with the technologic work on rare metals. The mill will be adapted to the application of several of the processes which have been evolved in the Bureau's Denver laboratory. None of the details will be made public until the results are officially announced through the Bureau.

With the impetus given to the radium industry by the organization of the institute, and the cooperation of the federal government, we may expect to witness a few disreputable promotions by unscrupulous parties who will prey on the public through promises of fabulous returns on a small investment. Such a development must result in inevitable loss to investors, besides reacting unfairly on a few commercial undertakings of a legitimate character. It would seem to be in order to sound a warning against promiscuous investment in "radium" companies.

Concentration in the Joplin District

Representatives of the Bureau of Mines have recently spent some time studying the methods of mining and concentrating zinc and lead ores in the Joplin district, Missouri. Their observations are contained in *Technical Paper* 41. Joplin concentrating mills have been the subject of much criticism, particularly in regard to their inefficiency in treating the finer portions of the pulp. The jig practice is regarded with more favor, and is generally conceded to give excellent results on the local ores. "In the concentration of the fine material the operators of this district have, until the last few years, been rather behind in their methods, but are beginning to realize the importance of saving the finer mineral particles formerly wasted in the discarded sands. There are still several mills in the district that have no concentrating tables or sand jigs, no attempt being made to avoid the losses mentioned." Even in those mills where tables are used, little care is exercised to get good classification.

The loss in concentrating lead ore is not as great as in the case of zinc. "The average recovery of blende from the ore in milling is about 60 to 65 per cent. When the heavy losses in smelting zinc ores are included, the total loss in the production of zinc, from the ore in the mine up to the commercial product, reaches nearly 50 per cent." Tests were made on the feed and tailings from different mills, with the following results: On ore from "sheet-ground" mines, rougher jig feed contained an average of 3.57 per cent zinc, and the tailings 1.36 per cent. From other than "sheet-ground" mines, rougher jig feed contained an average of 2.79 per cent zinc, and tailings 0.84 per cent. The average table feed in a number of mills was 5.28 per cent zinc, and tailings 1.57 per cent. The losses in different sizes of tailings are shown below:

	Dry weight grams	% of whole	Zinc assay %	Loss %
Oversize to ⅜ in.	2426	15.225	0.77	12.256
⅜ in. to 3 mm.	7579	47.565	0.96	47.608
3 mm. to 1.5 mm.	3817	23.955	0.76	18.946
1.5 mm. to 0.46 mm.	1659	10.412	0.71	7.678
Undersize through 0.46 mm.	453	2.843	4.55	13.512

An excuse for the tendency to sacrifice efficiency for capacity in the Joplin mills and mines, is found in the system of royalties which are imposed on lessees and sub-lessees. Royalties to owners of the ground run from 10 to 15 per cent of the gross output of mineral, and this is sometimes increased by the practice of subleasing, to 20 or even 30 per cent. The system results in waste in both mining and milling.

Strike Situation in Michigan

The latest reports from the copper country of Michigan are to the effect that normal conditions are gradually returning. Some demonstration of violence is still made by the strikers, particularly toward men coming into the district in search of work. These violations of the strike injunction have caused the arrest of a large number of strikers. The mine managers have issued a statement through the Copper Country Commercial Club to the effect that striking miners will be allowed to return to work, provided they have not engaged in riots and disturbance. An 8-hour day is to be established for underground work by the first of the coming year. As a result of these overtures work is being resumed and production is increasing. The Calumet & Hecla is employing nearly its normal quota of labor, and shipments of rock to the mills are being made to the extent of 7,000 tons daily. No shortage of labor is anticipated, as many of the iron miners are now available following the closing of some of the iron mines for the winter.

Settling the Fume Question in California

A state investigation of the differences between California smelters and farmers is under way, in accordance with an act of the last legislature which provided an appropriation of $5,000 for the use of a commission charged with the determination of damage by smelter fumes to stock and vegetation. This commission is composed of Dr. W. F. Snow, secretary of the state board of health, Dr. Charles Keane, state veterinarian, and Dr. J. A. Cook, commissioner of horticulture. The first work of the commission is being done in Shasta county, where a number of smelting plants have been closed.

Dr. J. A. Holmes, director of the Bureau of Mines, visited the affected districts in Shasta county on his recent return from Alaska. It is understood that he regards the situation as more hopeful of settlement than at any time past. General prosperity of the farmers in the smoke belt has diminished their antipathy toward the smelters, and complaints are fewer and less bitter.

Company Reports

Quarterly reports of the activities of the large copper companies for the third quarter of 1913 show no marked changes in conditions compared with the first half of the year. The Utah Copper Co. produced 23,884,467 lb. copper from 2,035,391 tons of ore averaging 1.2459 per cent copper at a cost of 9.068 cents per pound. Crediting the net miscellaneous earnings, the cost would be 8.187 cents. The low grade of the ore treated was responsible for the comparatively higher cost. Both the Magna and Arthur plants are in good condition, and handled an average of a little over 22,000 per day for the quarter.

The Ray Consolidated Copper Co. produced during the third quarter of this year 12,969,120 lb. copper from 575,190 tons of ore averaging 1.72 per cent copper. The average mill recovery was lower and the operating cost higher than for the previous quarter, owing to shortage of water during the summer months, and to some important improvements in the mill. The average cost of copper produced, after allowing for smelter deductions and applying the earnings of the Ray & Gila Valley R. R., was 10.155 cents per lb.

The Chino Copper Co. produced 15,187,003 lb. copper during July, August and September, 1913. The total amount of ore treated was 507,650 dry tons, of an average grade of 2.23 per cent copper. The extraction averaged 66.98 per cent; the average grade of concentrate produced was 14.46 per cent copper. The cost per pound of copper, after allowing for smelter deductions, but without crediting miscellaneous income, was 8.41 cents. If miscellaneous earnings are credited, the cost is only 8.08 cents.

The Nevada Consolidated Copper Company's operations the third quarter of the year resulted in the production of 15,835,563 lb. of copper. The amount of ore treated was 813,153 tons, 93 per cent of which was from the pits and 7 per cent from underground workings. The grade of the ore was 1.53 per cent copper, which is lower than that for last quarter. The cost per pound of copper produced was 10.09 cents, which is much higher than that for the preceding quarter, viz., 8.95 cents. The grade of concentrate produced is decreasing on account of the character of the ore now being milled. Experiments are being carried on looking to a cheaper and better method of treating the low-grade slimes which are produced in milling, and which reduce the grade of concentrate.

The annual report of **Camp Bird, Ltd.**, for the fiscal year ended June 30, 1913, shows that the mine at Ouray, Col., is able to supply ore for only 25 out of the 60 stamps. The gross amount of ore treated was 30,012 dry tons. The content of the ore treated averaged as follows: Gold, 0.958 oz. per ton; silver 3.83 oz. per ton:; lead, 1.42 per cent; copper 0.21 per cent. The saving of the gold amounted to 93.79 per cent. Of the total value extracted, there was obtained by amalgamation 49.28 per cent; by concentration 43.91 per cent; and by cyanidation 6.81 per cent. In addition to the above tonnage of ore treated, the cyanide plant handled also 1745 tons of impounded tailing, from which it is estimated that $4,937 was recovered.

The Camp Bird company's operations at the Santa Gertrudis property in Pachuca, Mex., resulted in bringing up the capacity of the new mill to 25,000 tons per month. During the fiscal year there were treated 263,554 dry tons of ore of an average value of $13.08 per ton. The recovery amounted to $11.70 per ton, or 89.41 per cent. The old Guadalupe mill was not run during the year; but 761 flasks, or 28.6 tons of quicksilver were recovered from the old patio, yielding a net profit of $17,335.

The Non-Ferrous Metal Market

Dullness has characterized the non-ferrous metal market during the period since our last report. Most of the prices have suffered a decline, and in cases where business has been good, the competition has been keen and prices have suffered. The dullness in some metal industries in this country, and the weakness of the London standard metal market have both been reflected in the domestic market.

Copper.—This market has been decidedly weak, and competition for orders has been keen. This has resulted in open cutting of prices, until unusual concessions were made. Business in Lake copper has fallen off until quotations are strictly nominal at 16@17 cents. Electrolytic is quoted at 15.25@15.30 cents.

Tin.—Prices have declined in the face of a fair statistical position of this metal. Considerable business has been transacted in a retail way, but the general tone of the market has not been strong. November tin in the domestic market is quoted at 39½ cents.

Lead.—This market has been fairly active, with sellers competing for such business as is in sight. The last available quotations are 4.15@4.20 cents, St. Louis, and 4.30@4.35 cents, New York.

Spelter.—This market has declined during the month. Business has been light and producers have made concessions to get orders from consumers. The last quotations are 5.05@5.10 cents, St. Louis, and 5.30@5.35 cents, New York.

Other Metals.—The aluminium market has shown dullness and little business has been transacted. The New York price for No. 1 ingots is 19¼@19¾ cents. Prices for antimony range from 6.50 to 7.75 cents for various brands, with a fair business in sight. The quicksilver market is steady with no material change in prices. New York quotes $38.50 per flask of 75 lb. The domestic price at San Francisco is the same as for New York, with special prices for export trade.

The Iron and Steel Market

November has been a month of falling prices and sharply decreasing productions, the flow of orders having undergone a great diminution.

At the close of October the steel finishing mills were operating at an average rate of 80 per cent of full capacity; at the close of November operations were at an average of not more than 60 per cent, and prospects were that the rate would drop to 50 per cent before the end of December.

The report of a month ago suggested that enough had been developed to indicate that the tariff of October 4, 1913, was not directly the chief influence in the price decline. It is now clearly established that the tariff has had practically nothing to do with iron and steel prices, which have been falling because there was insufficient demand and therefore sharp competition for orders. It may be that the tariff is in part responsible for the general slowing down in business which has extended to steel, but that is really an altogether different factor.

The iron and steel market is passing through one of its periodic liquidations, and the nature and extent of such movements is well understood. The readjustment starts by orders becoming less than shipments, until eventually the mills must seek new business in order to maintain full operation. This results in price cutting and the open market declines. The existing contracts are on a basis below the open market, and when the market declines to or below the level of a given contract the buyer refuses to take any more deliveries under it without a price readjustment. This forces the mills to seek more new tonnage and then, unless there is artificial control of prices, the market declines rapidly. In the whole history of the American steel trade there is no case on record of prices being artificially maintained for the entire time from one period of activity to another. What has occurred at times has been the artificial maintenance of prices for a period, during which contracts would thus be kept alive and mills would be content to operate upon such business as was afforded at the controlled price level.

The most variable element in these periodic liquidations and readjustments is the time element. In that which followed the panic of October, 1907, the period of controlled prices lasted about 15 months, the actual liquidation and readjustment occurring during a few months following. The next time, in 1910-11, the period of control was less clearly separated from the period of active readjustment, the one merging into the other.

The present readjustment is sharply distinguished from its predecessors by there being no artificial control. The market is proceeding to the inevitable low point, not far from the cost of production, with relative rapidity. According to theories formerly promulgated in the trade, the element of time rather than of price reduction is the controlling one in forcing a revival in demand, but those theories have fewer adherents than formerly. If such a theory is correct, the price decline will not soon be followed by recovery, with forward buying and advancing prices, but there are many who urge that the

governing element is not time, but price, and that if a low point is quickly reached, the recovery will be correspondingly early. An excellent opportunity is about to be afforded to test the validity of the latter theory.

The total decline in finished steel prices now averages about $4 per net ton, from the high point of last winter and early spring. The decline thus far represents about 60 per cent of the advance which had occurred from the previous low point, which fell late in 1911 and was the lowest average level in the finished steel market since 1898, when prices reflected the condition of there having been an industrial depression of five years' duration, with wages and supplies very low.

Pig Iron.

Although pig iron experienced an extensive decline during the first seven months of the year, the decline proved insufficient to meet present conditions, and the recovery which began in August came to an end about the middle of October, since when the market has been falling rather rapidly. During the first seven months of the year the average decline was about $3 per ton, while August and September showed an average advance of 30 to 35 cents. Since the middle of October there has been a further decline of nearly 75 cents, so that the market is now about $3.35 a ton below the level at the beginning of the year, which was substantially the top level in the advancing movement of 1912.

The pig iron market is almost absolutely stagnant as to inquiries and orders, representing stagnation rather than a normal dullness. In the summer there was a moderate volume of forward buying, due to the temporary stiffening in the market, and as deliveries are still coming in, and no forward buying is contemplated, orders and inquiries are far below normal at the moment. The actual consumption appears to be fairly large, although there has undoubtedly been some decrease. The current market stands as follows, with most of the quoted prices representing the ordinary asking figure and being susceptible of shading in the event of interesting orders being negotiated: No. 2 foundry, f.o.b., Birmingham, $10.50; No. 2X, delivered, Philadelphia, $15.50; No. 2X, f.o.b. Buffalo furnaces, $13.50; No. 2 foundry, delivered, Cleveland, $14.25; No. 2 foundry, f.o.b. Chicago furnaces $14.75; at valley furnaces (90 cents higher delivered Pittsburgh) Bessemer, $15.00; basic, $13.00; No. 2 foundry, $13.50; gray forge, $13.25; malleable, $13.75. Ferromanganese is $50 for English and $49.50 for German, f.o.b. Baltimore.

Steel.

As was the case in October, the unfinished steel market has been made not by important transactions in the open market, but by the readjustment of contract prices, agreed to by the sellers rather than allow the consumers to make inquiry in the market. On this basis we quote the market roundly at $20.50 for billets, $21.50 for sheet bars and $26 for rods, f.o.b. maker's mill, Pittsburgh or Youngstown.

Finished Steel.

Since last report bars, plates, shapes and sheets have declined from $1 to $2 a ton. Nearly all products are weak, and present quoted prices represent what would be done on ordinary small lots, while in the case of attractive orders the prices would probably be shaded quite materially. Quotations are f.o.b. Pittsburgh unless otherwise stated:

Plates, tank quality, 1.25 cents.

Shapes, 1.30 cents.

Steel bars, 1.25 cents, base.

Iron bars, 1.45 cents, Pittsburgh; delivered Philadelphia, 1.27½ cents; f.o.b. Chicago, 1.15 cents.

Wire nails, $1.60, base; plain wire, 1.40 cents, base.

Sheets, blue annealed, 10 gage, 1.50 cents; black, 28 gage, 1.95 cents; galvanized, 28 gage, 2.95 cents; painted corrugated, 28 gage, 2.15 cents; galvanized corrugated, 28 gage, 3.00 cents.

Merchant steel pipe, ¾ to 3 in., 80 per cent off list.

Steel boiler tubes, 3½ to 4½ in., 69 per cent off list.

Standard railroad spikes, 1.55 cents, Pittsburgh; 1.60 cents, Chicago.

Button head structural rivets, 1.85 cents.

Cone head boiler rivets, 1.95 cents.

Cold roller shafting, 62 per cent off list.

Carborundum Litigation

The United States Supreme Court has denied the application of the Carborundum Company for a writ of *a certiorari* in the case of the Electric Smelting & Aluminum Company versus the Carborundum Company, which had been formerly decided in favor of the former company by the United States Circuit Court of Appeals for the Third Circuit. This decision was given in some detail in our issue of April, 1913, page 203. This terminates this patent law suit of many, many years.

Refractories Manufacturers' Association

The recent Philadelphia meeting of the Refractories Manufacturers' Association was a very enthusiastic affair and well attended, considering the fact that all the members had to come from a considerable distance.

This Association is doing great work in the matter of standardization. Heretofore, the brick made was almost any size that the manufacturer wanted to make them. Now all the forty-five concerns which belong to the Association, manufacture the same sizes, both in silica and clay.

Colonel H. D. Savage, of the Ashland Fire Brick Company, of Ashland, Ky., is the president of the Association; Mr. J. E. Lewis, of the Harbison-Walker Refractories Company, of Pittsburgh, Pa., is the vice-president; Mr. J. H. Cavender, of the American Refractories Co., of Chicago, is secretary-treasurer.

The executive committee consists of Mr. J. L. Green, of the Laclede-Christy Clay Products Company, of St. Louis, Mo.; Mr. C. S. Reed, of the Chicago Retort & Fire Brick Company, Chicago, Ill.; Mr. E. S. Hitchens, of the General Refractories Company, Olive Hill, Ky.; Mr. D. D. Davis, of the Davis Fire Brick Company, Oak Hill, Ohio; Mr. T. E. Thomas, of the Niles Fire Brick Company, Niles, Ohio; Mr. J. H. McFeely, of the McFeely Brick Company, Pittsburgh, Pa.; Mr. George H. Diack, of the Queens Run Fire Brick Company, Lock Haven, Pa.; and Mr. C. H. Claiborne, of the Union Mining Company, Mt. Savage, Md.

The American Blower Company, of Detroit, Mich., have purchased the entire Air Washer interests, including patent rights, of the McCreery Engineering Company, formerly of Toledo, Ohio, and later of Detroit, Mich.

Gas Engines.—The Mesta Machine Company of Pittsburgh, Pa., has issued illustrated Bulletin 1 on "Mesta Gas Engines," built in sizes from 350 hp upwards for any class of service and for any fuel gas, such as producer gas, blast-furnace gas, coke-oven gas or natural gas.

Alunite has been discovered in Arizona and Nevada and the deposits have been investigated by the Geological Survey. No prediction of the value of the deposits is given, but they are considered as warranting further prospecting and development.

Electrolytic Coating of Iron and Steel with Lead.—The London *Electrician* of September 26 states that Lead, Ltd., has acquired a process devised by Mr. Sherard Cowper-Coles for the coating of iron and steel with lead electrolytically. It is stated that by this process lead can be economically deposited up to a thickness ⅛ in., and that the method is suitable for protecting iron and steel from corrosion, and for the lining of pipes, tubes and chemical vessels for containing corrosive liquors. Steel plates coated by this process are much smoother than the ordinary terne plate, and there is no reduction in tensile strength or ductility. The process can also be used for the coating of earthenware and wood and for the protection of ornamental ironwork. The cost is stated to be low.

Present Status and Future Prospects of the Metallurgy of Zinc.

A symposium before the New York Sections of the American Electrochemical Society and the American Institute of Mining Engineers.

The joint meeting of the New York sections of the American Institute of Mining Engineers and of the American Electrochemical Society held on the evening of November 20 in the Engineering Building was well attended, there being some 200 members present, and the carefully arranged program deserved the interest which was shown.

There were three papers. Mr. George C. Stone presented a very careful review of the improvements which have been attained in the metallurgy of zinc with the time-honored retort process; Mr. W. R. Ingalls discussed the possibilities and limitations of electric zinc smelting, while Dr. J. W. Richards dealt with electrolytic zinc, having stepped in at the last minute to cover this phase of zinc electrometallurgy, as a promised paper by Dr. Victor Englehardt, of Germany, on this subject, had not arrived in time.

Mr. Lawrence Addicks, chairman of the New York Section of the American Electrochemical Society, presided during the presentation of the three papers. Mr. L. D. Huntoon, chairman of the New York Section of the American Institute of Mining Engineers, presided during the discussion.

Improvements in the Retort Process

Mr. **George C. Stone,** of the New Jersey Zinc Company, in reviewing in a most interesting manner the development of the retort process, emphasized that while no spectacular or radical changes had been worked in the metallurgy of zinc as in the metallurgy of iron and copper by the introduction of converters, yet the progress had been as great with zinc as with the other common metals.

Three essential facts limit the possible apparatus for the production of zinc.

First, the temperature at which zinc is reduced by carbon is much above the volatilizing point of the metal, and it is, therefore, always produced as a vapor.

Second, to obtain a merchantable product the vapor must be condensed at a temperature well above the melting point of the metals. The temperature of the condenser must be close to the boiling point of zinc (920°) nearest to the retort and very much lower at the other end, because the condensation temperature depends on the concentration of zinc in the vapor and is lower the more dilute the vapor. The vapors are rich in zinc when they enter the retort and low in zinc when they leave it.

Third, the reaction by which zinc oxide and carbon monoxide are changed into metallic zinc and carbon dioxide is reversible. The reduction of zinc proceeds only until there is a certain ratio of carbon dioxide of zinc in the atmosphere and then stops, and in order to effect complete reduction there must be an excess of carbon always present.

These three essential facts control the type of apparatus to be used. It must be a closed chamber. It must be heated externally because the gas from the heating fuel must not come in contact with the charge. As the mixture of ore and coal forming the charge is a poor conductor of heat the size of the chamber containing it must be small in at least one direction or the heating will be too slow. The charge must, at all stages, contain a large excess of carbon to reduce at once all CO_2 formed. The condensers should be as close to the charge as possible and as small as will do the work in order to make it possible to maintain the proper heat gradient in them. The condensers must be kept within proper limits of temperature, diminishing towards the exit.

Mr. Stone explained in some detail why he questioned the statement commonly made that his spelter furnace is extravagant in heating, and then summed up the progress made in recent years as follows:

Forty or fifty years ago the typical Belgian furnace contained from fifty to ninety retorts of about 0.8 of a cubic ft. capacity each. At the present time the furnaces used in the natural gas field contain from 500 to 700 retorts of about 1.4 ft. each, an increase in capacity of about 1300 per cent. Furnaces containing 1008 retorts are in operation doing satisfactory work; this is an increase of about 2400 per cent in size and capacity. The increase in capacity of the spelter furnaces is, therefore, about the same as that of the modern coke furnace making pig-iron over the old charcoal furnaces.

Mere increase in size, however, does not amount to much unless accompanied by increased efficiency. In this respect the zinc furnaces show well. In the iron furnaces there has been no improvement in the proportion of metal recovered, the old furnaces having given almost theoretical results. With zinc the improvement has been very marked. In 1844, at the Vieille Montagne works, the recovery was about 62 per cent, and even as late as 1880 a recovery of 75 per cent was considered good work. To-day 87 per cent is only fair, and many works can show runs of long periods averaging 90 per cent and over.

The fuel consumption has been decreased largely mainly by the adoption of regenerative gas-firing. Even in direct-fired furnaces the decrease is large. Comparing furnaces only using the same kind of coal, where the hand-fired furnaces twenty years ago required 3, 3½ and even 4 tons of coal per ton of ore, well-equipped gas furnaces using the same coal, and working the same class of ore, now take only 1½ to 1¾ tons.

Labor has also been largely reduced. The old Belgian furnaces required five days' labor per charge, two men working twenty-four hours continuously, and an extra helper on the day shift only. As these furnaces worked only about a ton of ore they required five days' labor per ton. At present all twenty-four-hour work is done away with, and not over 1½ days' labor is required per ton.

In 1844 fifteen to twenty days was the usual life of a retort. Now good retorts made in hydraulic processes last from thirty to forty days, and at that more are replaced because they are filled up than because they leak.

Mr. Stone believed progress would continue to be made rather on the lines of improvement in the present retort process than by the adoption of radically different ones. He finally discussed briefly three different new types of processes which have been proposed, namely, the wet, electrolytic, and electrothermic.

One main reason for the lack of success of the wet process is the difficulty of dissolving the zinc. Zinc blende usually contains a large amount of iron and when roasted the zinc and iron combine to form zinc ferrite, which is practically insoluble in either acids or alkalies under economic conditions. The purification of the solution is also expensive and rather troublesome.

Electrolytic zinc of excellent quality has been made by one firm for several years, but they have never increased the original plant. It is generally believed that this plant is only possible in connection with other operations carried on by them. Besides other difficulties of wet processes Mr. Stone said that electrolytic processes have the trouble of large power requirements.

Electric smelting of zinc ores is seriously limited in this country by the fact that in most mining districts electric power cost is high. Further, there are condensation troubles and other difficulties.

Mr. Stone concluded that the present types of furnaces can be much improved. Fuel can be saved by better design and proportions of regenerators; better arrangement of the ports and decreasing radiation losses by more thorough lagging of the parts of the furnaces from which radiation is purely wasteful. A determination of the best size and shape of retort and condenser for different ores, and the choice of such as suit the particular ore to be worked instead of using one size and arrangement for all, will give improved results in recovery and output. A careful study and application of the principles on which ores should be mixed offer a wide field for improvement.

Electric Zinc Smelting

Mr. **W. R. Ingalls'** discussion of the possibilities of the electric furnace for zinc smelting was very careful and conservative. Mr. Ingalls said he was neither a pessimist nor an optimist with respect to electric zinc smelting, but an agnostic. He agreed that electric zinc smelting was no longer in the laboratory stage and referred to some enlargements now being made in the electric zinc furnace plants of Norway. He thought reckless talk had held back the development of electric zinc smelting.

At considerable length Mr. Ingalls discussed the well-known condensation troubles, blue powder being obtained instead of spelter.

The fire-smelting process is a two-stage intermittent process of heating up and distillation. The electric smelting process is intended as one continuous operation. This distinction causes important differences.

The condensation troubles originate at least partly in the furnace itself. Mr. Ingalls distinguished between two kinds of blue powder—meltable and non-meltable. The fact that the condenser is too cold may account in part for meltable blue powder, while non-meltable blue powder is due to impurities in the vapor to be condensed—if it contains carbon dioxide, volatilized lead, dust and other impurities. Oxidized lead is often found in large extent in blue powder.

As to the different electric furnace designs for zinc smelting, Mr. Ingalls could not see any great difference between them. He thought there was not much in electric zinc smelting at present beyond visions and ideas. There were few figures and the subject was attracting attention chiefly on account of its mystery.

Mr. Ingalls then compared the different items of cost of electric smelting and retort smelting. But few of the items in question were known besides the cost of electric power as against coal. He thought the consumption of 1200 kw hours per ton of ore to be plausible though hardly yet an accomplished fact. As to the comparative cost of retorts and electrodes, he maintained his former statement that nobody can yet say which will be the weaker. If the cost sheet of the Trollhättan plant was refigured by inserting American for Scandinavian prices the cost of the application of the process would be found to be absolutely prohibitive.

Mr. Ingalls concluded that electric zinc smelting was still very much in its infancy and that at the present time all we could say was that we did not know.

Electrolytic Zinc

Dr. **Joseph W. Richards** then discussed electrolytic zinc. In order to show the variety of processes which have been devised in this field he gave a brief outline of the contents of Guenther's German book on the subject. He then took up recent developments with reference to a lecture by Dr. Victor Engelhardt which was published in abstract in our January issue, 1913 (Volume XI, page 43).

While Dr. Engelhardt considers the mere refining of commercial zinc analogous to copper refining, to be economically absurd, Dr. Richards thought it might be possible, under favorable conditions. It would, of course, be necessary to keep the solution clear. It can be kept free from iron by bubbling air through it, whereupon it should be filtered clear. Such an electrolytic refining process might be particularly advantageous if carried out for electroplating purposes.

Engelhardt had stated with regard to direct processes using soluble anodes containing zinc that many attempts had been made since 1880 to utilize the energy due to the solution of the anode and to prepare anodes containing various zinc compounds mixed with coke or coal powder with and without pressure with various binding materials, etc. All such processes have proven a failure. Dr. Richards thought, however, that packing the ore around an insoluble anode (instead of incorporating it in it) was a possibility. He then commented briefly on Engelhardt's description of the latest Siemens & Halske process with insoluble anodes and finally discussed broadly the scheme of sulphate roasting.

Dr. Richards thought that sulphate roasting was essentially a misconception. One gets a dilute solution of sulphuric acid which must be dispensed with after all the zinc is gotten out, and the sulphuric acid produced is a source of trouble.

The correct way to work is, according to Dr. Richards, to roast the ore to oxide as nearly as practical. It is then leached with sulphuric acid and electrolyzed in a double compartment electrolytic cell, zinc being deposited on the cathode and sulphuric acid is then used for leaching another quantity of roasted oxide ore whereby the acid is neutralized. The zinc sulphate thereby obtained is again electrolyzed, etc. In this process, therefore, the sulphuric acid passes continuously through this cycle. It is formed in the electrolysis, neutralized in leaching, again formed in electrolysis, etc. The first supply of sulphuric acid is obtained from the roast furnace gases. This principle is used in a plant at Butte, Montana.

Discussion

Dr. **C. F. Chandler** opened the discussion with a characteristic little speech. When the chairman, Mr. Huntoon, called him up Dr. Chandler said: "My old pupil gets even with me. He knows I am not prepared, and that is why he calls me up." He then spoke briefly on two occasions in his career when he got into connection with zinc—the first time, 50 years ago, in a poison-murder case, where he had trouble to get zinc free from arsenic, and the second time, later on, in a lawsuit on electrolytic zinc.

Mr. **Woolsey McA. Johnson** was then asked to give a review of his work on electric zinc smelting at Hartford, Conn. With respect to Mr. Ingalls' statement that a power consumption of 1200 kw hours per ton of ore was plausible and pretty good, Mr. Johnson said this would not be the minimum, but rather the maximum consumption he would consider acceptable. With a small furnace running with 25 per cent efficiency he had usually an energy consumption of 1400 kw hours per ton of charge, but he had also done it at 900 or 1000 kw hours. With a larger furnace he was sure to get a much higher efficiency and reduce the energy consumption correspondingly.

As to the electrode consumption this was 6½ to 7 pounds of electrode carbon per 1000 kw hours, and he figured the cost of electrodes was considerably less than the corresponding expense for zinc retorts.

Mr. Johnson thought that the field of the electric furnace was not so much for plain zinc ores as for complex sulphide ores and by-product recovery must be considered if we want to see what electric zinc smelting can do and will do.

About roasting, Mr. Ingalls had expressed the opinion that the cost would be about the same for electric smelting and for the retort process. But Mr. Johnson pointed out that with the electric furnace it is by no means necessary to dead-roast the ore. Roasting down to 4 per cent of sulphur is about right for the electric zinc furnace. Sometimes in the Hartford plant they add green ore to get a matte. They do not want it to be too metallic and, therefore, add sulphur. This point should make a big difference in the cost of roasting for electric smelting and retort smelting. There is a vast metallurgical difference between a roast to half of 1 per cent and a roast to 4 or 6 per cent, both with respect to capital cost of the roasting furnace and with respect to the cost of roasting. Nine-tenths of all roasting troubles is caused by removing the last 40 or 50 lbs. of sulphur. This point also has a distinct bearing on the sulphuric acid plant for the reason that a richer gas is made. Mr. Johnson said that they are now building a three-ton furnace at Hartford, and he would be glad to show it to visitors. "Zinc comes from Missouri, and you have to show people."

Mr. **Alfred H. Cowles** spoke of some experiments made by him many years ago on electrolytic zinc. One difficulty which he encountered was the high resistance of the zinc sulphate solution; this was overcome by putting the electrodes close together by the use of two drums. The zinc deposited on the wires could be easily scraped off. The process used was cyclic with respect to sulphuric acid as described in Dr. Richards' paper.

The chief difficulty which was encountered was the melting

together of the finely divided zinc deposit. They tried for half a year all kinds of fluxes, but were unsuccessful until they found that it could be quite easily done with zinc chloride. It would be worth while to try this in the blue powder difficulty. The use of zinc chloride is not patented. Mr. Cowles pointed out that in comparison with aluminium production about one-seventh of the amount of energy is required to deposit a pound of zinc as is necessary to deposit a pound of aluminium.

Mr. **H. A. Wentworth** asked whether any unmeltable blue powder was formed at the higher temperature before the fumes reached the condenser at all. Mr. Ingalls said that he had no evidence to that effect and thought that the blue powder was formed in the condenser.

Dr. **J. W. Richards** referred to Mr. Ingalls' statement that there was no great difference between different electric zinc furnaces. While this might appear to be so at first glance, Dr. Richards insisted that there was a very great and important difference if the matter was investigated more in detail and the differences in the furnace designs had a great deal to do with the condensation problem. In the DeLaval type of furnace the fumes acted as resistor, and were thereby overheated, which increased the condensation difficulties considerably. On the other hand, a furnace using a buried arc or smothered arc was better, but it had zones of high temperature, and within these zones there was liability of evaporation of materials like silica or silicon which would later cause condensation troubles. But, in the whole, this type of furnace had certain advantages over the type in which the fumes acted as resistor. Finally, in resistance furnaces of the Clerc type or the Queneau type the conditions seemed to be quite different and probably more favorable.

Mr. **E. G. Spilsbury** emphasized that zinc is produced in the retort furnace not so much by contact with carbon monoxide gas, but by actual contact with elemental carbon itself. He recited some unsuccessful experiments which he had made some years ago on the use of carbon monoxide gas as reducing agent.

Mr. **Woolsey McA. Johnson** said that they had overcome their condensation troubles largely. Their average condensation factor (the ratio of weight of zinc in spelter produced to the sum of weight of zinc in flue dust plus ladle skimmings plus blue powder plus spelter) at Hartford was 75 per cent. They produced 1 to 5 per cent of blue powder and 15 to 20 per cent of flue dust. He emphasized that a sharp distinction should be made between blue powder and flue dust. One form of blue powder was particularly troublesome, namely, that due to the reaction between zinc and SO_2 or sulphur from the matte, and referred to his Canadian patent allowed February, 1913, in which a metallurgical description of "sulfidized chemical" blue-powder and ways of preventing its formation is clearly given.

After a vote of thanks, proposed by Professor Kemp and seconded by Mr. Stoughton, to the speakers of the evening and to the Committee of Arrangements, the meeting adjourned. It was followed by an enjoyable informal social gathering in the rooms of the Institute of Mining Engineers.

The Wellcome Photographic Exposure Record and Diary for 1914 has just been issued by Burroughs, Wellcome & Company, of London and New York. (Bound in red cloth, price 50 cents.) As in former editions, a very large amount of useful information is condensed in this little and neat pocket guide to photography. Exposure, development, toning, fixing, printing, the various processes of production in warm tones and colors, and the methods of dealing with errors of technique are explained, particular attention being directed in the 1914 edition to the green and blue toning, and the production of various colors by development and other methods.

The cost of copper produced by the Isle Royale company in 1912 was 11.89 cents per lb. The average yield from the ore was 15.4 lb. refined copper per ton. The ratio of concentration at the mill was 93 tons of ore into one ton of mineral which yielded 71.43 per cent fine copper.

The Passivity of Metals

A Symposium Before the British Faraday Society

For the meeting of the Faraday Society, held on November 12, 1913, in London, a symposium of highly interesting papers on the subject of the passivity of metals had been arranged.

We herewith give abstracts of the papers presented. An account of the general discussion which followed will be given in the letter of our London correspondent in our next issue.

The following abstracts are taken from advance copies of the papers kindly sent to us by the secretary of the Faraday Society, S. F. Spiers, 82 Victoria St., London, S. W.

Introduction Into the Passivity of Metals

A paper forming a general introduction to the discussion had been prepared by Dr. **George Senter,** who pointed out the marked divergence of opinion which exists with regard to the interpretation or explanation of the experimental results on the passivity of metals.

The more important phenomena which have to be accounted for on any theory of passivity are as follows: The passive state is best known in connection with iron, cobalt, and nickel, but most, if not all the metals may be made to assume this state under suitable conditions. Passivity may be induced by immersing the metal in an oxidizing agent such as nitric or chromic acid, and in solutions of alkali hydroxides; also by anodic polarization in a great variety of electrolytes.

When the force producing passivity is removed the metal as a rule returns to the active state, slowly at first and then more rapidly, in some cases almost instantaneously. When the surface of a metal in the passive condition is scratched or rubbed, it regains its activity at once. Treatment with acids, cathodic polarization with hydrogen, and raising the temperature all favor return to the active condition.

A metal becomes much more positive (more "noble") when it changes from the active to the passive condition, and does not dissolve when the equilibrium potential is reached. On further increasing the anodic potential the metal may dissolve as a more highly oxidizing iron (e. g. chromium), or the current may be employed mainly or entirely in discharging anions. The rise of potential on anodic polarization is usually slow at first, and then there is a sudden rise of considerable amount, the metal almost becoming insoluble.

The nature of the anion has considerable influence. Thus the ions of oxidizing acids, such as NO_3 and ClO_3, favor passivity, while Cl and Br have the contrary effect.

Three theories of passivity have obtained a greater or less degree of recognition:

(1) The *oxide-film theory* of Faraday. (*Phil. Mag.*, 1836, vol. 9, p. 53.)

(2) The *valency theory* of Krüger-Finkelstein (*Zeit. Phys. Chem.*, 1902, vol. 39, p. 104) and W. J. Müller (*Ibid.*, 1904, vol. 48, p. 577); according to this theory passivity phenomena are due to the change of a metal to a "nobler" modification.

(3) The *reaction velocity theory* of LeBlanc (*Zeit. f. Elektrochem.*, 1900, vol. 6, p. 472, and 1905, vol. 11, p. 9). In its most general form this theory states that passivity phenomena are due to slow rate of change (electrochemical or purely chemical) at the anode. In this form the reaction velocity theory is little more than a statement of the facts, and several special hypotheses have been put forward as regards the mechanism of the retardation. Thus we have:

(a) The *oxygen charge hypothesis* of Fredenhagen (*Zeit. Phys. Chem.*, 1903, vol. 43, p. 1; 1908, vol. 63, p. 1) and Muthmann and Frauenberger (*Sitzungsber. der Kgl. Bayrischen Akad.*, 1904, vol. 34, p. 201). The cause of passivity is to be sought in the slow rate of reaction between the anode and the oxygen liberted there, with the result that the anode becomes charged with the gas, or, alternatively, a metal-oxygen alloy is formed.

(b) The *anion discharge hypothesis,* according to which the main change at the anode is not the formation of metallic ions but the discharge of anions: the slow reaction of the dis-

charged anions with the metal produces passivity. The views of Sackur are based on the assumption of a primary anion discharge.

(c) The *hydrogen activation hypothesis* of Foerster (*Abhandlungen der Bunsen Gesellschaft*, 1909, No. 2), and of Schmidt (*Zeitsch. Physikal. Chem.*, 1911, vol. 77, p. 513), according to which the normal condition of iron is passive and it only becomes active under the influence of a catalyst. According to Foerster the catalyst is hydrogen; according to Grave (*Zeit. Phys. Chem.*, 1911, vol. 77, p. 513), a pupil of Schmidt, it is hydrogen ions.

(d) The *retarded ion hydration hypothesis* of LeBlanc (Lehrbuch der Electrochemie, 5th edition, p. 285).

Each of these different theories is briefly considered and discussed by the author. He finally concludes that in all probability it will eventually be found that no one theory will serve to account for all the phenomena of passivity.

Thus both LeBlanc and Foerster, although in the main favoring other explanations, recognize that the formation of insoluble films on the electrodes plays a part in some cases of passivity.

The hydrogen activation theory, which is perhaps most in favor at the moment, is undoubtedly valid to some extent, as the effect of nascent hydrogen in restoring the activity of passive metals is beyond dispute.

Whether iron entirely free from hydrogen is necessarily passive, as Foerster and Schmidt assume, would, however, appear to be still open to discussion. In particular, the activating influence of the free halogens (e. g., iodine in potassium iodide) is difficult to reconcile with this view, as it would appear that the metal in the absence of hydrogen is capable under these circumstances of sending charged ions freely into solution, which is contrary to the theory.

On the other hand, it should not be difficult to devise a plausible explanation of the *sudden* changes from the active to the passive state and vice versa on the basis of the hydrogen activation theory, whereas these changes are difficult to reconcile with any of the other theories. It is not easy to understand, for instance, how such sudden changes could occur on the basis of the ion hydration hypothesis favored by LeBlanc.

Anodic and Cathodic Retardation Phenomena

A paper by Dr. **G. Grube,** of the Institute of Technology of Dresden, Germany, discusses anodic and cathodic retardation phenomena and their bearing upon the theory of passivity.

There used to be a general tendency to assume that both the formation of metallic ions in the anodic dissolution of a metal and the transition from the ionic into the metallic state in cathodic deposition were processes which took place at practically infinite velocity. It was the merit of M. LeBlanc to have pointed out that, with certain metals, the passivity finds its explanation in the circumstance that these metals emit their ions of lowest valency into the solution at a greatly restricted velocity.

That the fundamental view of the theory of passivity of LeBlanc is correct, that it to say, that both anodic dissolution of a metal and cathodic deposition of a metal may be retarded phenomena, has within recent years been established by many experimental researches.

The author deals especially with the second part of Foerster's passivity theory, according to which the small reaction velocity marking the anodic dissolution and the cathodic deposition of passive metals does not alone suffice to focus the whole domain of activation and passivation, and that we are hence forced to assume a co-operation of positive and negative catalysts on the electrode.

The following important question thus suggests itself as concerning the problem of passivity: In how far is the velocity of reaction, and hence the variation of the potential in anodic and cathodic processes, influenced catalytically by the occurrence of small quantities of foreign substances on the surface of the electrode?

The author gives a review of a series of experimental researches carried out by Foerster and his pupils during the last years.

In reviewing their results the author accentuates the following main points:

1. The cathodic deposition of the metals—in itself a slowly progressing reaction—can still more be retarded to a considerable extent by the simultaneous deposition of zinc or of hydrogen. The cause of these retardations is probably to be found in a change of the surface of the electrode, the change being characterized by the formation of unstable intermediate products of higher electrolytic solution pressure than the pure metals possess.

2. The circumstance that cathodic processes can materially be retarded by the occurrence of small quantities of foreign substances on the electrode lends essential support to the assumption that anodic processes may likewise be retarded by the formation of oxygen alloys on the anodes. In many cases, therefore, the cause of the anodic passivity will not be a mechanical closure (covering up) of the electrode surface by an oxide film, but the formation of an alloy consisting of the material of the anode surface and of oxygen, this alloy having a lower electrolytic solution pressure than the pure metal.

3. In the electrolytic oxidation of ferropotassium cyanide to ferripotassium cyanide a strong polarization was always observed when the electrodes were surrounded by films of ferrocyanide or of oxides. These strong polarizations were not found with platinum, gold, nickel, cobalt, and copper in alkaline solutions. It may, therefore, be concluded that, in this latter case, the passivity is caused, not by an oxide film, but by an oxygen alloy on the anode.

4. The retardations observed in the anodic discharge of the halogens are likewise to be ascribed to alloys of the platinum anodes with oxygen.

5. The anodic evolution of oxygen shows what we have to understand by the passivating oxygen alloys. We may imagine that the oxygen combines to a small extent with the anode material to form oxides, which in their turn form, together with the unchanged electrode material, solid solutions, and therefore, one-phase systems of continuously variable composition and likewise variable electrolytic solution pressure. The oxide film theory contrasts with this view, since according to that theory the oxide formed would mechanically cover and shut off the electrode surface so that oxide film and electrode would form a two-phase system. The oxygen-alloy theory of the passivity thus seeks the cause of the passivity in a change of the chemical properties of the anode material, while the oxide-film theory finds the cause in a change of the mechanical properties of the electrode surface.

The Electrode-Volume Constant-Sum Hypothesis

A paper giving a "review and interpretation of recent experiments which extend and elucidate the domain of the passivity of metals," by Dr. **D. Reichinstein,** of Zurich, Switzerland, develops a theory which is an extension of the views of LeBlanc that it is velocity phenomena which condition passivity. The principal question discussed is how the transition is taking place from the active state of a metal into the passive state.

Several series of experimental facts are first discussed with respect to chemical polarization of reversible active metallic electrodes and with respect to "negative depolarization" (when a material added to the electrolyte does not chemically react with it and diminishes the current intensity instead of increasing it).

The author then passes over to a discussion of direct experiments on the passivation of metals.

"Surprise has often been expressed at the fact that a metal turning passive does not, at higher current density, show at least that anodic solubility which belongs to it at small current density. Let a passive metal be dissolved anodically at a low current density a to the extent of 50 per cent, and let it liberate oxygen, so that its dissolution takes place at $a/2$ amp./sq. cm. of metallic surface.

"When we increase the current tenfold, the dissolution will not proceed at the rate of $5a$ amp./sq. cm.; it does not even take place at the former rate $a/2$ amp., but at a much smaller rate. How is this to be interpreted chemico-kinetically?

"Let us assume that a passive metal Me generates primarily oxygen. Two reactions are possible:

(1) $Me + O + 2H^{\cdot} = Me^{\cdot\cdot} + H_2O$.

(2) $O + O = O_2$.

"The commencement of reaction (2) is designated 'passivity.' The example teaches us that passivity is not a result of two competing reactions, for both of which the velocity would increase with increasing oxygen concentration, but the passivity is caused by two reactions such that the velocity of the one reaction decreases with increasing O concentration, whilst the velocity of the second reaction regularly increases with increasing O concentration."

The author introduces the conception of the "electrode volume." The contact of an electrode with the electrode is assumed not to be formed by the solid metal, but by a solution (or alloy) of all those substances whose ions are represented in the electrolyte. The volume which this hypothetical alloy occupies in the electrode is designated "the electrode volume."

Now concerning this the following important hypothesis is introduced: "The sum of the concentrations of all the constituents of the electrode volume is always constant, whether the current flows or not."

The electrode volume is always fully saturated, and any new constituent, possibly introduced by the electric current, ejects parts of the old substances.

When a metal electrode is treated anodically the oxygen introduced pushes the metal atoms out of the electrode volume —the same will happen on the cathode by the H_2 introduced— and the chemical equilibrium prevailing before the current was flowing will be destroyed. The oxygen now begins to react with the metal atoms of the electrode volume. Let this reaction be of the form:

$$Me, O + 2H^{\cdot} = Me^{\cdot\cdot} + H_2O,$$

where Me and O are respectively an atom of the metal and of the oxygen of the hypothetical alloy, and $H^{\cdot}$ concerns the electrolyte.

If a is the constant sum of the concentration of the constituents of the electrode volume, then the author shows that the following holds true:

When with a certain current density of the electrolysis the compensation process is so rapid that the oxygen concentration of the electrode volume does not rise above the value $a/2$ (which may be called the critical concentration), then there will be no reason to fear that another compensation process will set in, e. g., generation of oxygen (= passivity). When, however, the critical concentration is exceeded, the compensation process will necessarily be retarded, and the o concentration will increase rapidly until a new compensation process sets in.

The case of solution of gold in cyanide is discussed in some detail.

Photo-Electricity and Passivity

A paper by Dr. **H. Stanley Allen,** of the University of London, deals with "the photo-electric behavior of iron and the theory of passivity."

The author points out it is not unusual to find that investigations in one branch of science serve to throw light on problems connected with some other subject. The author's research was undertaken in the hope that the study of the physical phenomenon of photoelectricity might assist in elucidating the century-old problem of the nature of "passivity."

The photoelectric effect here referred to is that which is known as the Hertz-Hallwachs effect. It consists in the loss of a negative electric charge or the acquisition of a positive electric charge by an insulated metal when the surface of the metal is eliminated.

The author's researches show that "iron which is chemically active shows large photo-electric activity, while processes which render iron passive greatly reduce this activity. These facts are in good agreement with the theory, which as well as the oxidation theory must be attributed to Faraday that the cause of passivity is to be found in the condition of the gaseous layer at the surface of the metal."

Hydrogen Theory of Passivity

A paper by Professor **Gerhard C. Schmidt,** of the University of Muenster, Germany, contrasts the oxide theory according to which metals are normally passive and passivity is due to a coating of oxide or oxygen on the metal, with the hydrogen theory according to which the metals in question are normally passive and become active only in the presence of dissolved hydrogen acting as a catalytic agent.

The latter view has recently been strongly supported by the author's pupils Grave and Adler on the basis of experimental investigations, but Flade has criticized the result as non-conclusive. The author now describes further experiments made by W. Rathart.

These experiments deal with the effect of a hydrogen charge on the potential and activity of a metal and further give the proof that a metal is rendered active by diffusion of hydrogen ions. This proof is given both for iron and chromium.

In this proof advantage was taken of the fact that when hydrogen is liberated electrolytically at one side of a piece of iron foil it rapidly diffuses through and alters the e. m. f. on the other side which is under test.

The author's views of the phenomenon of passivity agree with those of Grave. "Just as water or other liquids do not boil, even when their vapor pressure is equal to that of the superincumbent pressure, unless a catalyst such as air is present, so the metals which can be passivated dissolve rapidly only in the presence of a catalyst. The most important of such catalysts is hydrogen, since it is dissolved readily and in large amount by metals. Whether other gases are also catalytically active cannot at present be stated, since they are only absorbed to a very small extent by metals. Just as relatively small amounts of air initiate rapid evaporation in large quantities of liquid, so a small quantity of hydrogen can activate large amounts or iron, nickel, and chromium. It might be supposed that the hydrogen, like air in the case of boiling, forms a nucleus around which the ions of metal collect. Further investigation must decide whether this hypothesis is in accord with the facts."

Passivity and Reaction Velocity

In a paper on "the phenomena of passivity" by Professor **Max LeBlanc,** at the University of Leipzig, Germany, the author first reviews the oxide theory of passivity and numerates the objections which have been raised against it. He concludes that it is evident that the hypothesis of an impervious layer, which covers the surface of the metal, thereby protecting it from contact with the electrolyte, is open to the most serious objection.

Professor LeBlanc then takes up the study of polarization measurements, when employing a system, metal—a not too dilute solution of a metallic salt—metal (e. g., copper-copper sulphate solution—copper). It was expected that no appreciable polarization would take place in the case of short-time electrolysis with moderate current density. While this expectation was fulfilled in some cases it was not fulfilled in others. How can this be explained?

In considering the action at the anode, it is conceivable that primarily no metallic ions are formed, as is generally supposed, but that the electrical process consists of a discharge of negative ions and the chemical process of an interaction between the non-electrified radical and the metal. The potential difference would, according to this view, depend upon not only the concentration of the negative ions, but upon that of the separated non-electrified radical, and consequently upon the velocity with which the latter reacts upon the metal.

In a similar manner the cathodic polarization can be explained. When the current passes, the free ions separate, which

must, in the state of equilibrium, according to the law of mass-action, always be present in addition to the hydrated ones. Consequently a diminution in concentration and then polarization follows, owing to the insufficiently great speed of the formation of free from hydrated ions.

An analogous explanation may be made for the phenomena of passivity. With passive metals ionic hydration occurs so slowly at the anode (a certain amount of solution nearly always takes place), that the concentration of free ions and, therefore, the potential difference, becomes so great in a short time that visible separation of the negative radical or of oxygen can result.

From this point of view we must henceforth understand by passivity phenomena all such as result from chemical polarization. Passivity is not the exception, but the rule, and it is not connected with oxygen-containing anions, as was supposed, but it takes place also with chlorides and other solvents, and especially it can also take place at the cathode.

Anodic Oxygen and Oxide Films

A paper by Professor **E. P. Schoch,** of the University of Texas, discusses "the mechanism of anodic reactions and the behavior of iron and nickel anodes."

The mechanism of any anodic reaction is considered to be the direct attack of the anode by the anions. From Guenther Schulze's experimental work on the electrolysis of phosphate, borate, etc., solutions with an aluminum anode (*Ann. d. Phys.*, vol. 28, p. 789) the conclusion is drawn that a film, or cover, over the surface of an anode will hinder or prevent the direct access to the surface of the metal of complex anions and will bring about the formation of oxygen ions from them. The latter pass on to be discharged on the surface of the metal.

This gives all the facts necessary for the explanation of the chemical change of any anode in any electrolyte. Briefly this explanation may be stated as follows:

"With a clean anode surface the anions come in direct contact with the metal, react with it, and give up their electrons; the rapidity of the reaction and the final products obtained will vary greatly with different metals and different anions.

"The product may be so rapidly formed and so rapidly dissolved that other anions may have unhindered access to the surface. Again the reaction may be slow, or the product may dissolve slowly, or the latter may be insoluble, or it may be hydrolyzed into an insoluble compound, so that other ions may find their progress to the anode more or less obstructed—a condition that may naturally be brought about in almost any case by a sufficiently great current density.

"Many complex ions which are prevented by the obstruction from reaching the surface of the metal will dissociate into an equivalent number of oxygen ions, which reach the surface of the metal more readily.

"Upon discharge these oxygen ions may form the oxide of a metal (as with copper in sulphate electrolytes), or they may form partly an oxide and partly oxygen (as with platinum in sulphate electrolytes), or they may react so slowly that the discharged oxygen still in contact with the metal becomes gradually more condensed as more oxygen ions are forced into the same area with a greater potential difference until finally gaseous oxygen, or perhaps some higher oxide of the metal is formed.

"In the absence of any oxide skeleton the layer of oxygen, on thickening, will soon allow bubbles of gas to escape—as is probably the case with platinum, iridium, etc., in which the oxide formed is within the body of the metal.

"But when a non-conducting oxide skeleton is formed the layer of gas may accumulate to a relatively great thickness as the oxide skeleton grows, and this may prevent more or less completely the passage of any ions to the electrode.

"Finally, if the oxide is oxidizable to a conducting (per-) oxide, then a suitable concentration of oxygen produced by its discharge under a sufficiently great potential will product this peroxide and oxygen gas will then be evolved from the new surface of the peroxide.

"The actual existence of this latter state of affairs has been beautifully demonstrated by E. Mueller (*Zeit. f. Electrochem.*, vol. 13, p. 133) who showed that a copper anode in sodium hydroxide solutions forms hydrated cuprous oxide as long as the current density is small enough; but with a larger current density the anodic potential becomes more noble, copper peroxide is visibly formed on the anode, and then oxygen is evolved from the surface of the latter. It appears that the "oxygen-producing" discharge of ions on peroxide anodes is a rapid reaction, because no great hindrance is observed even with great current densities.

"This beautiful demonstration by E. Mueller of the relations between active and passive copper indicates that the phenomena which characterize the attainment of passivity by some metals in certain electrolytes appears to be nothing but the regular behavior of anodes under the existing conditions."

Professor Schoch has reached the same conclusion from his study of the behavior of iron and nickel anodes in various electrolytes. (*Zeit. f. Phys. Chem.*, vol. 58, p. 301; *Am. Chem. Journ.*, vol. 41, pp. 203 and 235; *Jour. Phys. Chem.*, vol. 14, p. 739.) A review is given of the experimental facts which are then explained by the general theory of anodic reactions outlined above.

"No special attempt was made to state in what particular form the oxygen obtained by the discharge of oxygen ions is present on the surface of iron or nickel anodes, that is, whether it is present as free oxygen or as oxide. To the writer there do not appear to be any facts which establish one view over the other. The assumption of a free oxygen film is perhaps a trifle simpler than the assumption of a formation of an oxide, but the possibility of the presence of an oxide has not been disproven."

Professor Schoch finally discusses briefly the hydrogen catalysis theory.

"Possibly the best experimental evidence for a decision between the hydrogen catalysis theory and the theoretical views we have here advanced in the following.

"A chromium anode that has become passive with reference to the formation of bivalent ions does not become active again to form trivalent ions, but does become active again to form chromation.

"An iron anode that has been made passive with reference to the formation of bivalent ions does not become active again to form trivalent ions. However, when electrolyzed in acetate, or oxalate solutions the electrode becomes active again (that is, the metal dissolves again extensively) at potentials considerably below those (more noble!) at which it has ceased to be active before and at which it is usually passive in other electrolytes. The same thing is true of nickel (*Jour. Phys. Chem.* vol. 14, p. 719; *Jour. American Chem. Soc.*, vol. 30, p. 1737).

"In all three instances above the activity at the lower potential is due to a reaction which consumes oxygen and thus lays the metal surface bare. This is plainly the case in the formation of chromations. In the other examples the activity of the iron or nickel reappears when the potential is low enough to oxidize the acetion or oxalation, and when the oxygen is removed to oxidize these substances, then the metal surface is exposed and is subject to the attack of the anions."

Passivity and Electrolytic Valve Action

A paper by Dr. **Günther Schulze,** of the Reichsanstalt, deals with the relation of passivity to the phenomenon of electrolytic valve action. The author emphasizes the remarkable analogy between the phenomena of valve action and passivity which follows from the assumption that valve action is caused by an insulating film and consequently a gas layer, while passivity takes place if the film shows metallic (electronic) conductivity together with a limited or negligible solubility in the electrolyte.

The consequence of the metallic conductivity of the film is that it cannot grow much above molecular size and must be almost undetectable by optical or mechanical means. The oxygen developed at its outer surface covers it more or less and protects it against the electrolyte when it has been formed

Thus the current may be decreased below the critical forming value without restoring activity.

The behavior of iron in H_2SO_4 of different concentration is of special interest. In dilute sulphuric acid it shows passivity, in concentrated acid valve action. In intermediate concentrations an unstable valve action is succeeded by stable passivity. In dilute solution the anions SO_4 may form an oxide of iron by the aid of the water. In concentrated solutions the small amount of water present is not available for this reaction, as it is bound to the sulphuric acid itself. Therefore, ferric sulphate must be formed. This insulates and causes valve action.

"Therefore I hold: Valve action is caused by an insulating skin of ultra molecular thickness with a thin gas layer within it; passivity by a conducting skin of molecular thickness with a molecular gas layer upon it."

Electric Furnaces, Their Design, Characteristics and Commercial Application

By Woolsey McA. Johnson and George N. Sieger

Buildings for Electric Furnace Plant

The character of building depends entirely on the stability and certainty of operation. For a small experimental plant a wooden shack 16 ft. x 32 ft., 10 ft. to eaves, 16 ft. to ridgepole, costing $140, is good enough.

When the work has a reasonable certainty of continuing two or three years an edifice like the new furnace building of the Continuous Zinc Furnace Company, 32 ft. x 60 ft., built of heavy braced wooden framework, with a roof supported on strong wooden trusses, sides, ends and roof covered with No. 26 galvanized sheet steel, is neat and serviceable. This building was designed by F. E. Waterman, architect, of Hartford. The total cost without architect's fee was $698. This did not include concrete floor, which was afterward laid on a floor of ashes and slag.

For a more permanent structure, a structural steel framework covered with galvanized sheet steel is to be recommended. This will cost nearly double what a similar structure of spruce framework costs. Its main advantage lies in the fact that it is fireproof.

For a permanent structure designed to last ten to fifteen years, a structural steel framework with curtain walls of red brick or hollow tile with a concrete roof covered with tar and gravel or covered with tar and gravel on a felt covering is ideal, but this will cost about 2.5 times what a galvanized-iron structure costs.

Rough figures per cubic foot for average conditions in the United States (and these are not far from wrong for any place as where lumber is cheap, supplies are apt to be dear, and where labor is low it is apt to be inefficient) are given below:

Kind of Structure.	Cost Per Moderate Height Per Cubic Foot.
Wood shack	$0.02 to $0.025
Galvanized sheet on wooden framework	0.30 to 0.035
Galvanized sheet on steel framework	0.60 to 0.075
Hollow tile walls steel framework	0.10 to 0.150

Concrete roof covered with tar gravel.

FIRE PROTECTION

In any electric-furnace plant, as in any metallurgical plant, while the building is in course of erection, it is good business to install a fire hose in a corner or a number of them in corners. These hoses should have a large pipe connection with a quick-turning shut-off valve to city main or system that has good water pressure back of it.

A safe rule is if one puts in 1-in. fire hose to connect it to an 1½-in. pipe. If one wishes to exhibit a great pretence of accuracy and scientific engineering, multiply the diameter of fire hose by $\sqrt{2}$ or 1.414 and then add 10 per cent, guessing at the final proper size for pipe connection. The main point is to have the pipe connection considerably bigger than the fire hose.

The fire hose with nozzle is coiled on a framework and placed in a box made of double walls of sheet steel or "asbestos-wood," and having space between inner and outer walls filled with mineral wool. In the bottom of these boxes is placed an electric heater of 500 watts capacity. If the door shuts tightly, and the heater is on, the fire hose will never freeze up. The nozzle should be securely fastened to the hose so that it cannot be taken off to furnish water for the masons or for mixing concrete.

Around the walls as a further fire protection, small chemical extinguishers using acid and soda ash or "squirt guns," using carbon tetra-chloride, may be provided. These are first rate for small incipient blazes. In case of a fire the first thing for the superintendent to do is to see that high-tension switches are thrown, so that water can be played without men getting "shocked," and then to save any note books and measuring instruments from damage by water or fire. The men will use hoses with zest, energy and dispatch, and the superintendent can smoke his pipe.

The Continuous Zinc Furnace Company has had twenty-seven fires in the past six years and none of them has caused damage amounting to more than $25. If it had not been for efficient fire protection the amount of damage undoubtedly would have been serious. With such a protection the only damage from fire can come when nobody is at the plant, as during the night when no runs are going.

FOUNDATIONS FOR ELECTRIC FURNACES

In view of the hazardous and uncertain element in electric-furnace work, especially at its inception, emphasis must be laid on the necessity of having foundations which are above all question. Test pits should be dug and a bearing value of ground measured by well-known engineering methods. Then a large factor of safety should be employed.

For a small furnace, weighing from 1 ton to 5 tons, a bearing value of 1000 lb. per square foot is a conservative assumption. In this case a hole 36 in. deep and 48 in. square can be dug, filled with concrete (a 11:2:55 mixture is all right). This will hold up 16,000 lb., unless the ground is soft on one side and hard on the other.

A cheaper way is to put in large blocks of masonry or of concrete from demolished furnaces and pouring a *wet* mixture of sharp sand and cement (1 to 1). Old wire, barrel hoops, old pieces of pipe thrown in tie the piece together and make a monolith.

When a larger furnace is to be supported an excavation 48 in. to 72 in. deep can be made. In such a case old girder rails tied together with corrugated iron bars in regular fashion and a whole mass of 1:3:6 concrete tamped in. Where special strain comes a mixture 1:2:4 can be used in the monolith.

This scheme is good engineering where a total load of 125 tons to 175 tons is to be held up. Fig. 1 shows this.

But when loads exceed 175 tons, there is a risk of uneven settling and piles must be used. A pile will hold up from 10 tons to 40 tons, according to size of pile and according to force of blow that pile-driver used can deliver. The pile should be capped with a steel hoop or "hat" of cast iron. Above the pile special reinforcing should distribute the upward forces in the form of an inverted pyramid. On top of this should come a platform of reinforced concrete.

The reason why piles are advisable is that the certainty of their holding up the load is as absolute as anything can be.

POWER FACILITIES

The authors believe that in general the power facilities should have 20 per cent. capacity in excess of the normal rating of furnace. There always comes a time in the operation of a furnace when it pays to force the furnace, for the reason that the furnace is cold and working slow, or because the ore is especially easy-working and a little forcing will increase daily tonnage. With this reserve power capacity above normal furnace demands the furnace will be operating all the while at its maximum commercial efficiency.

The great tendency in design of electric furnaces is to underrate them, either because of the fact that thermal efficiencies are usually estimated at too high a figure or because heat-balance sheets are made that are too optimistic. Moreover, the power-factor or load-factor rarely comes up to what is hoped for or the drop in potential from switchboard to electrode is greater than was originally estimated.

It may seem paradoxical with a 200-kw furnace to have a 250-k.v.a. transforming or generating apparatus, in view of the fact that any well-designed electrical apparatus is capable of being overloaded for short periods tremendously. But if we apply first principles of commercial science we will find that the following reasoning applies:

(1) Electric furnace processes if successful at all have a large margin of profit.

(2) That which makes them successful at all is the powerful metallurgical effect of the electric current.

(3) Therefore, the supply of this should be ample to carry operation along at a maximum rate of daily tonnage at will of the manager.

(2) Raising the voltage up or down by varying field voltage of electric generator or by cutting in or out resistance in field circuit.

(3) Where an electric transformer is used, use may be made of cutting out number of turns in primary or secondary winding or by changing either of these from series to multiple or to semi-multiple, or increasing or decreasing the magnetic reluctance of circuit.

(4) By using a water-rheostat or other variable resistance in series with the electric furnace.

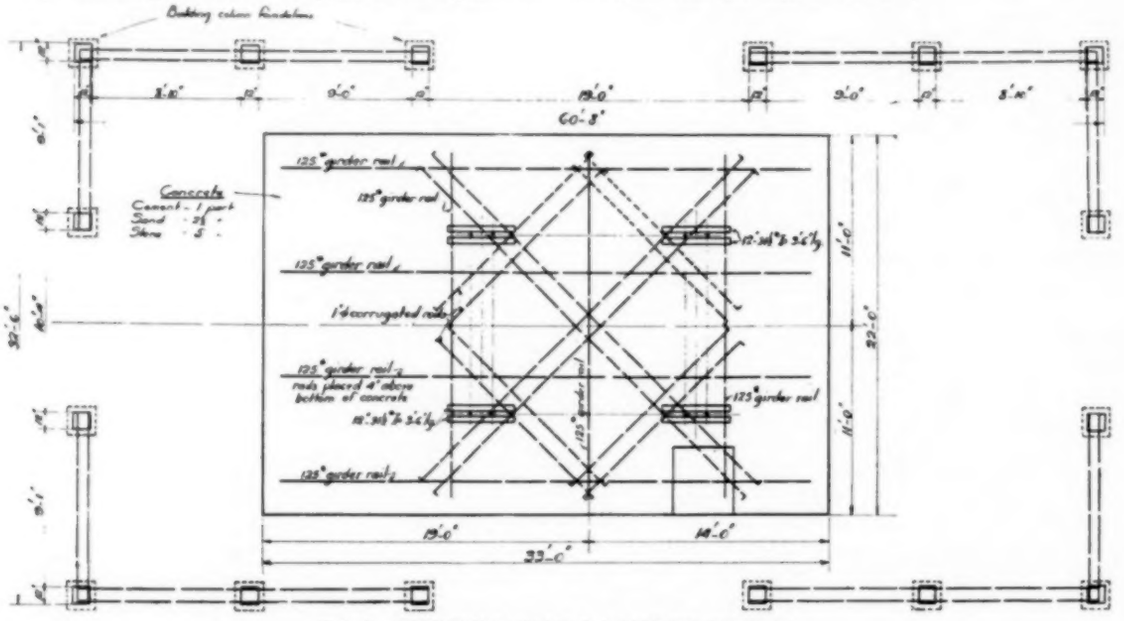


FIG. 1.—FOUNDATION PLAN FOR ELECTRIC FURNACE

VOLTAGE REGULATION

There have been few large electric furnaces built that operate at a constant voltage. The reason for this is the fact that the voltage depends on the resistance of the furnace and that this resistance depends greatly on the temperature. Substances that do not conduct cold at all conduct well at high temperature. To use a striking example, a 100,000-volt modern porcelain insulator *would break down at 25 volts at 1400 deg. C.* With this tremendous change from a perfect insulator to a good conductor, it is evident that the regulation of voltage must be under good control, for temperature is sure to vary from time to time.

Now, one way to do this is to have some automatic regulation of the length of circuit through which the electric current plays. The best example of this is seen in the classical steel furnace of Simens. As lately practicalized by Girod and Héroult in the modern electric steel-refining furnace, the voltage is kept nearly constant by moving the electrode automatically up or down by a relay-controlled motor. The relay is actuated by a device which depends for its motion on some function of the electric current.

When no such internal regulation is used, external voltage regulation is employed. This can come from the following scheme of design.

(1) Speeding up or down the prime-mover such as steam engine. Such a process can only be used well when one furnace is operated tied electrically to a generating set.

When doing electric zinc furnace work for the Lanyon Zinc Company in 1903 and 1904, one of the authors used this electrical layout.

An old 75-kw. Westinghouse alternator was belted to the line shaft of the power plant. The field of this was excited by an old Edison direct-current generator. The field current of the former could be controlled by one rheostat and field current of the latter by another rheostat so that any voltage from 300 to 1000 could be obtained from the alternator at a maximum of 125 kw. with a minimum of 45 kw.

This high-tension circuit was put directly on the electric zinc furnace which was of ore resistance type making an infusible residue, when a furnace was started up from the cold.

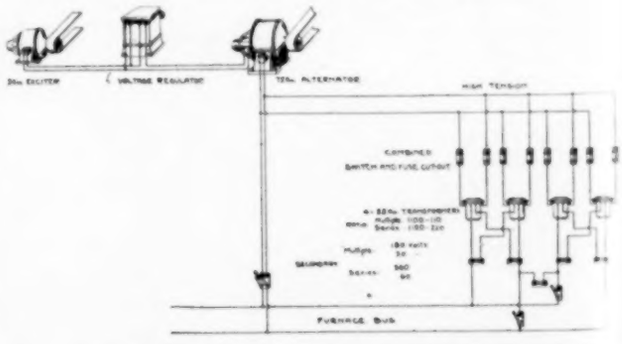


FIG. 2.—WIRING DIAGRAM FOR ELECTRIC ZINC FURNACE

As the voltage decreased this current was transformed by means of four 25-hp transformers (ratios 1100 to 220 and 1100 to 110). So a minimum voltage of 30 volts at 45 kw was at hand. The entire set gave thus a minimum of 45 kw at any voltage from 30 volts to 1800.

This scheme of electrical connections is shown in Fig. 2. It is a good layout for experimental work where plenty of surplus mechanical power is at hand.

ELECTRIC FURNACE TRANSFORMERS
LOW CAPITAL COST THE ESSENTIAL

Electric-furnace transformers are a special department of electrical engineering. In general cases a number of taps are brought out from the primary winding and by some switching device these can be cut in or cut out so as to make a varying transforming ratio and a variable voltage on furnace circuit.

A heavy double-throw switch on the secondary puts the secondary coils in series or in multiple, thereby changing the transforming ratio 100 per cent in one operation. This is the usual design.

Special devices such as choke coils in series or in shunt in the primary or some device that will change the magnetic reluctance of the transformer core may be employed.

The main consideration in designing an electric-furnace transformer is to make a practical machine and *one of low capital cost*. It is evident that with power at 2 mills to 4 mills per kw-hour little money is saved by increasing the electrical efficiency from 96 to 99 per cent.

Rather should one pay a few more dollars on the power bill and cut down hundreds of dollars on the interest and depreciation account, for in a new industry this figure the authors place at 20 to 35 per cent per year.

Good commercial design of an electric-furnace transformer has this commercial fact incorporated in itself.

The subject of electric-furnace transformer design is an important one and will be made the subject of a forthcoming separate article of this series.

The transformer plant should be either outside on a platform or in a separate building away from dust and smoke.

The design of a transformer station of an experimental plant, of course, is quite different from that of a regular furnace plant. Here the main point is to buy as little as possible and it is good business to waste power seemingly quite recklessly but really scientifically.

Any experimental plant should have on the high-tension circuit either an oil-switch or a "disconnecting" switch on pole—better both. In time of trouble these can be "pulled" and "juice cut off" so that repairs can be made in safety.

Fuses should be eliminated or of large capacity so that nothing but a prolonged short-circuit will blow them.

High-tension circuit breakers made *so as to break the current instantaneously* and placed so that all breaking of the furnace current is done on the high-tension side are the right things to install.

At the plant of the Continuous Zinc Furnace Company there

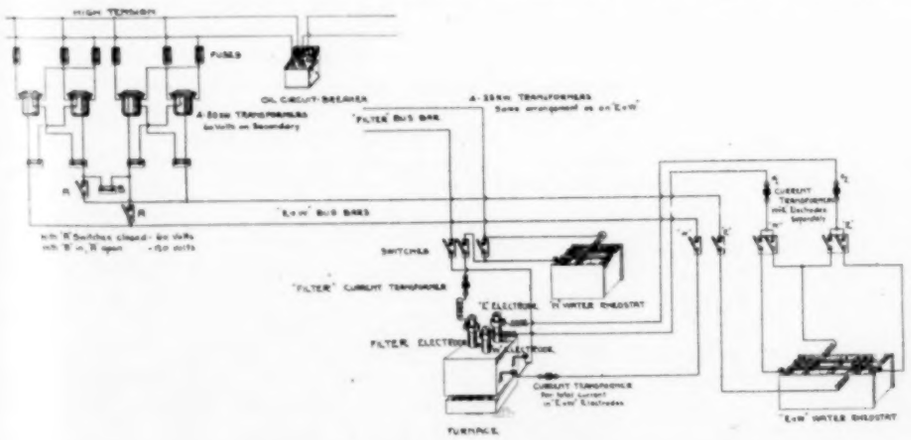


FIG. 3.—DIAGRAM FOR ELECTRICAL CONNECTIONS

are four 50-kva transformers on each phase. The voltage ratio is 2400 to 240 to 120 on each transformer. Each phase has the transformers arranged in two sets of two transformers in series.

The various voltages on the bus-bars are secured by connections made both on the primary side and on the secondary side.

Provision is made for connecting the sets together in parallel when 60 volts are desired, or in series when 120 volts is wanted the two transformers of each set are connected in parallel. 240-volt connection can be made by disconnecting the sets and putting the primaries of the four transformers of each group in multiple, and the secondaries of each coil in series.

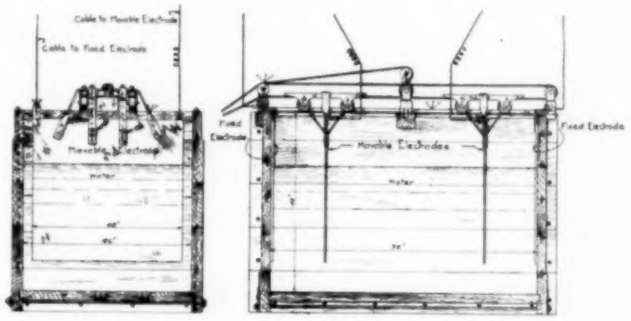


FIG. 4.—WATER RHEOSTAT

Intermediate voltages between zero and 60, or 60 and 120, 120 to 240, are obtained by the use of water rheostats.

Recently there has been installed an arrangement of disconnecting switches on the primary side so that the primary of each of the four transformers can be connected in series, whereby 30 volts on the secondary is obtainable.

Thus there is at disposal out of standard apparatus that will give 125 kva as a minimum at any voltage from 30 volts to 240 by imperceptible gradations.

Fig. 3 shows this.

WATER RHEOSTATS

A strong, durable and efficient water rheostat has recently been constructed at the Continuous Zinc Furnace Company's plant. The vat is 72 in. x 46 in. x 48 in. and is made of cypress wood 3 in. thick. All joints were splined and white-leaded. The whole is securely held together by the use of 5/8-in. rods running vertically and horizontally. The vat is designed to regulate the voltage of two furnace electrodes on the same phase.

Regulation is obtained by the use of three electrodes made of 3/8-in. perforated boiler plate 40 in. x 42 in. One of these, the neutral, is located in the center and is stationary. The other two are movable. The movable plates are attached to No. 1 Standard Fairbanks trucks, the tracks for same being 2-in. tee bars running across the top of the vat. The trucks are easily moved in either direction by means of ropes and pulleys.

Electrical connection is made to the boiler plates by first cleaning a small area of the plate and copper-plating it. A small piece of copper bar is then soldered to it and the connection is given mechanical strength by the use of a bolt. Fig. 4 shows this.

BARREL RHEOSTAT

An ordinary strong barrel (gin or whiskey) is used to an advantage in the construction of a cheap and serviceable water rheostat. In rheostats of this type used at Hartford, the fixed lower electrode consisted of a coil of 0000 copper wire about 3 ft. in length on which was placed an iron disk, 15-in. in diameter. A water-tight connection was made through the barrel by tightly screwing a brass nipple in the barrel, running the copper wire through this, solder being used for the seal. The copper wire was tinned and the inside of the brass nipple was tinned before pouring in the seal of solder.

The movable electrode was made of a perforated disk 1/2 in.

thick and 15 in. in diameter, connected to a 1-in. pipe as shown. To the upper end of the pipe a four-way was attached.

This served the double purpose of a means of support and for electrical connection.

The whole was suspended from a pulley by a ¼-in. hemp rope to the four-way.

Electrical connection to movable disc was effected by using a brass nipple and a flexible cable which was soldered to the brass nipple.

The idea of having copper wire enter through the bottom in a perfect seal, so as to avoid troubles of internal short-circuit, was suggested first by Mr. Cole, electrical engineer of the Johns-Pratt Company.

All rheostats should be placed outside, using soda ash, which is non-corrosive during all months save December, January and February, when common salt, which is slowly corrosive but which will not freeze, should be used as the solute to give requisite conductivity.

The big water rheostats will absorb without trouble 40 kw

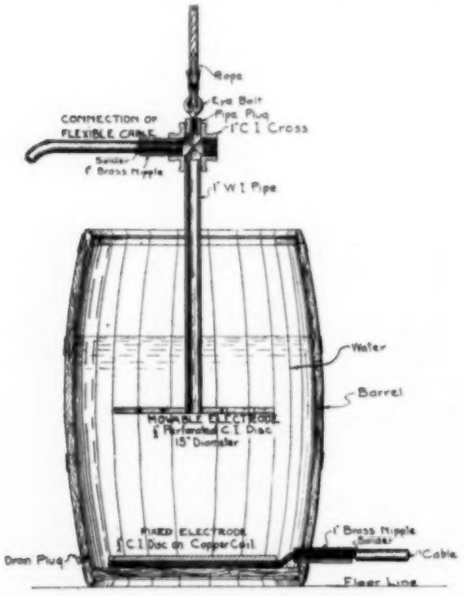


FIG. 5.—BARREL WATER RHEOSTAT

each. A barrel rheostat will absorb in constant service only about 5 kw.

Fig. 5 shows the Continuous Zinc Furnace Company's barrel rheostat.

BUS-BAR AND CABLES

Undoubtedly the cheapest way to carry electric-furnace current is by bus-bars. These sell at 2 cents to 4 cents above the price of ingot copper and will carry at least 25 per cent more than the same amount of copper in a bare cable and nearly double the amount of copper in an insulated cable. They can be bolted or clamped together. At the Hartford plant of the Continuous Zinc Furnace Company bus-bars are used for almost all switchboard work. We have a lot of right-angle bends and other short pieces, which are used just the way a plumber uses T's, L's, etc.

These are fastened together with malleable iron carpenters' clamps 4 in. to 6 in. in size. It is surprising how serviceable these ungainly circuits are, especially if care has been employed to see that the contacts are bright and smooth as all copper contacts should be. Bus-bar circuits are interlaced to cut down the self-induction and to raise the "power factor."

To the furnace, of course, flexible cables lead to electrode connection clamps. The lugs for these should be of high-conductivity bronze or forged copper. Where possible the cable should be pressed or forged into place or brazed into lug. Soldering is not the right way to do it, although it can be used. A special electric furnace cable with a core of asbestos rope of about perfect properties and characteristics is made by the General Electric Company.

Intercrystalline Cohesion of Metals

A second paper on this subject was presented by Dr. **W. Rosenhain** and Mr. **D. Ewen** at the recent meeting of the (British) Institute of Metals in Ghent, Belgium.

This paper is a continuation of a research undertaken in order to test by experiment the theory that the crystals of a metal are held or "cemented" together by thin films of the same metal in the amorphous or undercooled liquid condition.

In their previous paper the authors obtained evidence in favor of this theory from the behavior of metals when heated in a high vacuum. Since then the theory has received further support from the researches of Rosenhain and Humfrey on the tenacity and fracture of mild steel at high temperatures (Iron and Steel Inst., May, 1913). There it was shown that at high temperatures the type of deformation and fracture obtained by any but very rapid strains was entirely different from the type found at low temperatures. In the latter type both deformation and fracture occur principally within the crystals, while the boundaries act as stiffening and strengthening ribs; at high temperatures, on the other hand, the crystals remain almost undeformed while movement and ultimate fracture occur principally in the intercrystalline boundaries.

Detailed consideration of the observed facts shows that they can be readily explained by the amorphous cement theory—the cement being strong and hard at low temperatures, but, like all very viscous fluids, becoming soft and mobile at high temperatures.

An extension of this idea shows that at a sufficiently high temperature in all pure metals the "cement' must become so fluid as to possess no appreciable strength, while the crystals still retain a definite degree of strength and hardness. It follows that at such temperatures the crystals, even of the most ductile metals, should be capable of being pulled apart without undergoing deformation and the metal should exhibit extreme intercrystal brittleness.

This has been experimentally verified in the case of lead, tin, aluminium and bismuth, photographs of the strikingly intercrystalline fractures thus obtained being given in the paper —the brittle fracture of lead at 323 deg. C. (melting point 327 deg. C.) having an appearance like that of a brittle casting. These experiments show that the well-known brittleness of metals near their melting points is due to loosening of intercrystalline cohesion as indicated by the amorphous cement theory.

The authors deal with the suggestion that impurities concentrated in the crystal boundaries may affect these results and show that this objection cannot apply to their experiments and they strengthen this view by showing that a pure eutectic alloy (that of lead and tin) exhibits analogous phenomena.

Microchemistry of Corrosion

A paper by Dr. **Cecil H. Desch** and Mr. **Samuel Whyte**, entitled "The Michrochemistry of Corrosion. I: Some Copper-Zinc Alloys," was read at the recent meeting of the (British) Institute of Metals.

Experiments are described in which uniform areas of various alloys are corroded for a definite time under the influence of an electric current. The material dissolved by the electrolyte is analyzed, as well as the adherent metallic deposit. The surface of the alloy is also examined microscopically before and after the removal of this deposit. Only a few milligrams of material have to be used for each full analysis.

The alloys used in the first series of experiments are beta alloys of copper and zinc with and without the addition of tin or iron. Quenching from above the transformation point (470°) increases the corrosion, although the action is slightly retarded at first. Iron hastens corrosion, while tin retards it. The influence of the latter element, however, is chiefly mechanical, by means of the formation of a tough, adherent layer of basic salts, which serves as a protective coating.

The microscopical appearance of the corroded surfaces is characteristic.

The Iron Blast Furnace and the Characteristics of Its Fuels

Fundamental Conditions—Definition of Blast Furnace—Classification of Fuels Used—Essentials of Plant

By J. E. Johnson, Jr.

Iron is one of the most common elements in nature. Geologists state that it comprises 7 per cent. of the earth's crust and that there is strong proof that the whole interior of the earth below a depth of a few miles is composed of iron. The vast segregations which have gone on throughout geological time have resulted in concentration of iron to all degrees from the lowest up to 72 per cent. The concentration never exceeds this degree for the reason that iron is unlike most other of the useful metals in one important particular. It is seldom or never found native on the earth, except in the form of meteorites which have fallen from space.

This is in consequence of the great affinity of iron for oxygen, with which it will unite under almost all conditions of temperature and chemical environment found on the earth. This brings us to the fundamental fact in the metallurgy of iron which distinguishes it from all the other useful metals.

Its only ores are its oxides. The sulphides which are so important in the occurrence and metallurgy of other metals are not regarded as ores of iron, but only as ores of sulphur.

When the sulphur has been burnt off for conversion to sulphuric acid the iron takes up oxygen to replace the sulphur and becomes an oxide. These artificial oxides then become iron ores under certain conditions and with certain limitations which will be considered later.

But the use of iron sulphide as iron ore either by burning the sulphur off to waste, or directly in the furnace, as is so commonly done with the sulphides of other useful metals, is entirely unknown in the metallurgy of iron.

It is true that carbonates are important sources of iron in Europe, but their carbonic acid is not strongly held and before they are charged into the furnace are driven off by an oxidizing roast which converts the resulting ferrous oxide FeO to ferric oxide Fe_2O_3. This reaction takes place so easily that it will occur slowly under ordinary atmospheric conditions, but with the slight modification in that case that the ferric oxide takes up water of combination and becomes limonite.

More than 90 per cent. of the world's production of pig iron is derived from ores which occur in nature as oxides and the remainder comes from the roasted carbonates and the ferric oxide or "pyrite cinder" derived from the sulphides.

From the point of view of the blast-furnace, therefore, it is correct to say that the only ores of iron are the oxides.

The removal of the final traces of oxygen from iron is so difficult that it can commonly only be done with an expensive gas like hydrogen or methane or by solid incandescent carbon and as the latter is commercially so much more available we may say that industrially iron is reduced only with solid carbonaceous fuel.

Fundamentally then the metallurgy of iron consists of the reduction of the oxides through the use of solid corbonaceous fuel. This operation is carried on to the extent of a few hundred tons per year in the Catalan forge and to the extent of some sixty or seventy million tons in the blast furnace which, therefore, constitutes the foundation of the iron industry.

The blast-furnace is a shaft furnace, the "high-furnace" of France and Germany, and takes its English name from the fact that the air to support combustion must be forced into it under pressure because of the resistance to the passage of the combustion gases offered by its high shaft filled with material. It became, therefore, a "blast" furnace as opposed to a draft furnace, obtaining its air through chimney draft.

The blast-furnace is historically the outgrowth of an attempt to utilize the waste heat of a "low furnace" or Catalan forge so as to reduce the fuel required for the latter and is probably the first industrial application of what is called the counter-current principle.

This may be defined as a method of transmitting the properties of one mass of matter to another by causing the two masses to travel past one another in opposite directions in intimate contact. This process leads naturally also to continuous as opposed to intermittent operation, one of the great disiderata of industrial processes.

This practice has so long and so universally prevailed in the iron industry that it has caused us to overlook one of the most important facts concerning the blast furnace and one to which we shall have occasion to revert many times; namely that it is an apparatus in which we have merged two distinct processes. First, the reduction of iron oxide to metallic iron. Second, the melting of the reduced material.

The deep fire necessary for melting the "iron sponge" or reduced iron in a reducing atmosphere (without which it would instantly become oxidized again) was not unlike a gas producer in its construction and operation, and consequently discharged a hot gas rich in CO.

The utilization of this heat was the object sought by those early unknown pioneers who added the shaft above the low hearth or Catalan forge, increased the blast pressure to suit, and so transformed it to the blast-furnace.

The fuel economy desired was obtained to such an extent that the low hearth declined to a position of insignificance, although the change brought many other changes not contemplated by the pioneers, some of which were not to their liking.

Most of these must await discussion in a subsequent article, but one was vastly important because it made a radical alteration in the character of the product, and in the process. The original product was tough and pasty, could not be melted but could be "wrought" directly under the hammer, while the product of the improved furnace could not be worked under the hammer, either hot or cold, passing immediately from the brittle solid state to the liquid state in which it could be "cast" or poured. Hence the names of the two principal varieties of iron, "cast" and "wrought."

The production of the iron in the liquid condition contributed further to the continuity of the process since the iron could be allowed to gather in a pool at the bottom of the furnace instead of requiring to be worked out in lumps or "loups" with fire tools, when ready; and the subsequent introduction of a fresh charge of ore.

The shaft-furnace along with these advantages brought a requirement of far-reaching importance. The column of stock must be penetrable by a considerable gas volume at a reasonable pressure. This requirement is just as important to-day as it was in the early day of which I have spoken, though it must be confessed that our ideas of a "reasonable pressure" have undergone great changes.

This penetrability of the stock column may be seriously if not fatally impaired by either of two conditions.

First, the presence of tarry or sticky matter which would fill up the interstices of the charge and make it practically gas-tight.

Second, great fineness of the charge, either of the ore or of the fuel. Moreover, the fuel is subject to a certain solvent action by the gas as will be described later, and this is more active in proportion to the fineness of the fuel so that serious loss occurs if the latter be too fine.

The two principal fuels available, wood and bituminous coal, in their native condition both contain tarry material, but in very varying quantities. The vast majority of their varieties are, therefore, not adapted to use as furnace fuels without preliminary treatment to drive off the tar, so that this condition is industrially of paramount importance.

We can now combine all these necessary conditions to form a fundamental definition as follows:

The blast-furnace is a vertical shaft-furnace blown with air under pressure for reducing iron from the oxides and delivering it in a liquid condition. Its fuel must be not only solid but in lumps of considerable size and must be sufficiently free from tarry components to permit the passage of the gas resulting from the combustion in its hearth.

In addition to these fundamental requirements of a blast-furnace fuel there are others of nearly equal importance.

The three characteristics next in importance to those mentioned are first, purity, particularly freedom from sulphur and phosphorus. Second, porosity or ratio of surface to mass. Third, physical strength or resistance to crushing.

The reason for the importance of *freedom from sulphur and phosphorus* is that next to carbon these two elements exert the most important influence of any of the common constituents in steel both being destructive of quality if present to an extent greater than one-tenth of one per cent. If present in the pig iron of which steel is made they cannot be eliminated at all when certain processes are used and in the others only with increased difficulty and expense. In foundry irons phosphorus is not objectionable within certain limits, but sulphur is as much so as in steel.

The blast-furnace may be so operated as to remove the sulphur in the charge but only with increased cost for fuel and flux and with decreased output and in some cases with reduced quality.

This is not the case with phosphorus, virtually all of which present in the charge goes into the pig iron, in normal operation.

For these reasons sulphur in fuel is always bad and progressively worse as its amount increases, although its removal is possible.

Phosphorus in the fuel, on the other hand, may not be objectionable in any given quantity for certain iron, but if objectionable is prohibitive of the use of that material because its removal in the blast-furnace is impossible.

The importance of *porosity* of the fuel comes from the fact that the velocity of combustion of carbon in air is not infinite but limited. Only a limited amount of carbon per unit of surface exposed can combine with oxygen no matter how favorable the conditions and in order to secure the rapid combustion necessary to produce high output from a given size of furnace, much surface must be exposed by the fuel.

This cannot be done by reducing its size and so increasing the ratio of surface to volume and mass except to a very limited degree for reasons already explained, and high combustion velocities can, therefore, only be obtained with porous fuels.

The porosity also has a direct bearing on the volume of given weight of fuel, and this also will be shown subsequently to have an important influence on the operation of the blast-furnace.

Physical strength is important because without it no matter how large the lumps of fuel may be initially they are broken by the superincumbent weight of the charge column as the coke descends into the furnace and we have that condition of fineness of fuel which is so objectionable.

On account of the fundamental effects on the operation of the furnace and the quality of the iron caused by these chemical and physical variations in the fuel used, furnaces and the irons they produce are commonly and quite properly classified by the fuels used. These are in the order of their importance as follows: Coke, charcoal, anthracite, raw coal.

Coke

This subject is so important that large volumes have been written about it to which those in search of profound information are referred. It is only possible to give a bare outline here.

Coke is a light, very porous fuel produced by the distillation of mineral coal containing sufficient bituminous matter to cement together the whole mass at a certain temperature.

The distillation drives off nearly all the volatile matter and good coke consists almost entirely of fixed carbon and ash.

Coke is made in four ways, of which only two are metallurgically important.

First, in *heaps* or *meillers* on the ground. These heaps are covered with turf and dirt to prevent the access of sufficient air for complete combustion. The limited combustion permitted generates heat enough to drive off the volatile matter from the remainder of the coal and coke results, but the yield of coke per ton of coal is very small, the cost is high, and the valuable by-products are of course completely wasted in this process.

For these reasons this process is not in industrial use so far as known to me, though it is sometimes used to determine whether a given coal can be coked, and once during the last boom in the iron business it is said to have been used for a short time to secure a supply of coke for a furnace which could not get one otherwise, but the broad statement that the process is not in industrial use stands.

Second, *gas retorts*. One process of supplying illuminating gas for cities consists in distilling a high volatile coal in air-tight retorts with external heat. The coke is purely a by-product and is used in part for heating the retorts and in part for domestic and other non-metallurgical uses.

Its quality is very poor, being soft and friable owing in part to the kind of coal from which it is made and in part to the process of manufacture; for this reason it is not used in blast furnaces at all, its interest comes from the fact that the gas retort was one of the progenitors of the by-product oven.

Third, *bee-hive ovens*. These now produce about 75% of all the coke made in this country, and are, therefore, of vast commercial importance.

They consist of a bee-hive shaped structure of firebrick generally from 9 to 13 feet in diameter approximately hemispherical in shape. They have a floor of fire brick, a doorway about 30" wide by 36" high, in one side, at the bottom and a charging hole and smoke outlet combined in the center of the dome known as the "eye." These ovens are always built in batteries and frequently in double batteries with the doors in one row on the opposite side to those in the other. The space between them is filled with dirt level with their tops and a track runs over the top of each row to carry a "larry car" which delivers the coal to them. In the case of a double battery the track may be between the rows and the larry have a spout on each side to deliver the coal to either row desired.

The track for the larry is in good practice, always supported on brick pillars coming up from below the base of the ovens so as to eliminate the jar of the passage of the larry cars from the oven proper, as the life of the latter is seriously curtailed when it gets this continual pounding.

In operation the floor and walls of the oven are heated by maintaining a fire on the floor for several days. Then coal which will fill the oven to about 50 per cent. of its capacity is charged through the "eye" in the dome; and leveled off. The doorway has previously been built up with brick and clay almost to the spring of its arch which prevents the coal from running out. After the coal is leveled off, the rest of the door is built in the same manner; airtight except that two or three draft holes are left.

In a short time the heat of the oven begins to distil off the volatile matter which ignites and burns in the free space in the dome. This further heats up the coal and causes more volatile matter to distil off and burn in its turn. The whole mass becomes heated and fuses together in a plastic condition, its temperature being a dull red. The gas from the lower portion of the coal forces its way out through the plastic mass with the result that its structure is cellular to a high degree. John Fulton in his work on coke estimates that good cokes have a cell-space of about 45 per cent. of their total volume.

As a result of the decreasing combustion when the gas evolution diminishes, the temperature begins to fall and the mass of coke develops vertical cracks.

The process is not finished until the volatile matter is reduced to 1 or 2 per cent. As the amount of volatile matter coming off diminishes by degrees, the supply of air through the draft holes is cut off so that it may be, as nearly as possible, only as much as is needed for the combustion of this gas without leaving any air free to attack the coke itself. The air supply is then shut off altogether and after a few hours a spray-pipe is introduced through the top of the doorway and the surface of the coke thoroughly watered down. As little air as possible is admitted during this operation so as to

prevent loss of coke by combustion. After the oven is watered down the brick door is removed and the coke pulled out with hooks, rakes and other appropriate tools on to the platform in front of the oven where the watering-down process is further carried on if any of the coke is hot enough to require it.

As soon as possible after the oven is empty the brick door is reset and the oven charged again. This time the heat of the floor and walls of the oven is sufficient to ignite the coal in a short time without any preliminary firing.

The process is then carried on continuously, charging and drawing as rapidly as possible, because after the oven gets too cold it will not ignite the incoming charge of coat at all, or if the charge becomes ignited more of it has to be burned to waste to make up the heat lost by the oven by radiation while standing idle. The work of drawing is laborious and very hot. For this reason oven pullers are among the most difficult of labor forces to keep in regular supply in an industrial organization.

On account of the severity of the labor it is not customary to draw ovens on Sunday and this, with the necessity for having an oven charged immediately after drawing, necessitates two periods of burning. These are commonly 48 and 72 hours. This is regulated in large part by the size of the charge and in part by the supply of air admitted for combustion. The yield of coke in percentage of the coal charge is larger in good practice with 72-hour coke than with 48-hour, but the yield of coke per oven per day is considerably smaller with 72-hour than with 48-hour coke.

The coke made in the two periods differs to some extent in its physical characteristics. Seventy-two-hour coke is considered harder and, therefore, more desirable for foundry purposes.

The operation of coke ovens on this plan is so simple that it can be done by almost anyone, but in order to get the best results much skill and careful oversight are required. It is not impossible by superior management to increase the yield of coke per ton of a given coal from 60 per cent to 70 per cent of the coal charged in the same ovens. The matter of drafting is particularly important and a large percentage of the coke in the front of the oven may be burned up by improper draft.

It is considered by some coke-oven authorities very objectionable for all the air to be admitted at the relatively low elevation of the top of the doorway and all in one place because, owing to the delay of the air in becoming thoroughly mixed with the volatile matter as the latter distills, the air has an opportunity to attack the coke.

Owing to these considerations ovens have been built with an annular passage around the dome above the level of the doorway, this passage communicating with the oven proper by narrow radial slits in the brick work around the circumference of the oven and communicating with the outside air through a draft door. There is little doubt that better results can be secured with this construction if proper attention be paid to it, but the annular passage is likely to become filled with dirt and the narrow ports stopped up. These are difficult to clean out and when this is neglected the oven is no better than the plain oven which costs rather less. With first-class supervision the annular air-passage is probably a valuable feature, but where the operation of the ovens is left to careless and untrained men its benefit would be short-lived.

The volatile matter which distils off from the coal contains a considerable quantity of tar and during the operation this breaks up at the surface of the mass of coke, presumably, owing to the intensity of the radiation in that region, depositing fixed carbon of a beautiful silver-like luster on the face of the coke. If the watering-down is done without admission of much air, this will be cooled below its ignition point before air reaches it and the coke when drawn from the oven will have the silver luster by which its quality has been so widely judged in the past and is even yet to a considerable extent.

Fourth, *by-product ovens.* Even with best practice there is considerable loss of fixed carbon in coking in beehive ovens. Moreover the coal contains valuable by-products, especially tar and ammonia, which cannot be recovered in ordinary practice with the beehive oven. Also, the heat-value of the volatile matter is much in excess of that required for the coking operation. This is plainly indicated by the considerable flame rising from the "eye" of every oven in operation.

In the distillation of coal in gas retorts and its coking in coke ovens we have exactly the same fundamental operation carried on for two different purposes and for two different products. In the case of the retort the gas is the desired product, the ammonia and tar are recovered from it and sold. The coke is of little value.

In the beehive oven, on the other hand, the gas is distilled off from the coal much as in the retort, but the gas itself is thrown away or burned up, as are its valuable contents of tar and ammonia, the coke itself being the only product.

It was natural that in the course of industrial development attempts should be made to consolidate these two processes and to make simultaneously coke for metallurgical purposes and gas useful for heating and lighting, with the recovery of tar and ammonia. Such attempts began in Europe more than 50 years ago and owing to the greater cost of coal with the lower cost of technical supervision and labor the ovens resulting from these attempts have almost completely supplanted the beehive oven. In America the opposite conditions prevailing, beehive coke still constitutes three-fourths of the total output, but the by-product oven is making rapid strides and it is only a question of time when it will have displaced the beehive oven as completely as it has done in Europe.

In its essentials the modern *by-product* or *retort-oven* consists of a firebrick chamber about 18 inches wide, 6 feet high and 18 feet long, with flues on each side in which gas and air are burned. The ends are closed by metal doors lined with brick set against metal frames built into the brickwork of the oven. Instead of one opening for charging in the top there are three or more, and instead of the laborious operation of drawing by hand the finished charge is pushed out of the oven by a mechanically actuated ram carrying a sort of rectangular piston of the same general shape as the oven, but somewhat smaller. This is so powerful that it forces the whole mass of coke out at the opposite end of the oven.

There it is caught in a suitable grating or rack, and quenched with a heavy shower of water as quickly as possible. But no matter how quickly this may be done it has had access to the air and enough combustion results to destroy the silver luster, characteristics of good beehive coke. This makes the coke look black and lusterless. For this reason furnacemen in the past have frequently been prejudiced against it and in fact it was a matter of years before the builders of by-product ovens were able to produce coke of metallurgical quality equal to the best beehive. But that point has now undoubtedly been reached.

By-product ovens are built in long batteries parallel to one another and only separated by the air and gas flues on each side. The modified "larry" for delivering the coal to them runs overhead as in the beehive oven, but in handling the gas a vast difference exists. No air whatever is admitted to the ovens, the greatest pains being taken to preserve them absolutely air-tight. The gas is taken off through pipes which deliver to a common gas main, and all the gas is sent to a central purification plant where the tar and ammonia are removed and a portion of the gas is then returned to supply the heat needed for the coking process. The rest of the gas is available for either city lighting or industrial purposes.

In order to maintain the high temperature necessary to force the heat for coking through the firebrick walls of the ovens regenerators are commonly used to preheat the air for combustion and where regenerators are not used recuperators are substituted for them. The results obtained with these latter are not so economical of gas as those obtainable with the regenerators and the best practice tends more and more strongly to the use of the regenerator oven.

These regenerators are commonly built as the substructure

of the oven itself. Sometimes each oven has its own separate regenerator, and in other cases the regenerators are common to a whole battery of ovens.

The first two types of ovens to achieve commercial success in America were the Semet-Solvay and the Otto-Hoffman. The former designed uses horizontal flues, the latter vertical. In more recent years the Koppers oven or ovens of which it is the prototype have taken the lead over the other two. These have vertical flues like the Otto Hoffman oven of an earlier date, which they have largely replaced.

The yield of coke with the by-product oven is always higher than with the beehive. In fact the quantity of fixed carbon in the coke is greater in good practice than the fixed carbon in the original coal.

This apparent paradox comes from the fact that during the operation the tar and volatile matter of the coal are broken up by heat as described for the beehive oven, but unlike the latter the by-product oven does not permit the combustion of any appreciable fraction of the coke produced. The consequence is that the fixed carbon of the coal is augmented by the fixed carbon resulting from the breaking up of the bituminous matter without any loss through combustion.

A wider range of coals can be successfully coked in the by-product oven than in the beehive, though, on the other hand, some coals that can be successfully coked in the beehive oven swell to such a vast extent during the distillation of their volatile matter that it is impossible to push them from the by-product oven. These, however, are few in number.

Much attention has been paid to the best percentage of volatile matter for coking purposes in by-product ovens and its line of commercial development has been very different from that of the beehive oven. The latter has very generally been built as close to the mouth of the coal mine as possible with the object of avoiding transportation charges to the point of consumption of the volatile portions of the coal, and, as the yield from such ovens is relatively small, this is an important consideration.

With the by-product oven, on the other hand, the yield of coke is very much larger and moreover the gas is of great value, as above mentioned. In addition to this the by-product oven will do its best work on coal of a certain content of volatile matter approximately 25 per cent. This is naturally not always found in nature.

The volatile matter in coking coal covers a wide range. It is only from 18 to 20 per cent in the excellent coals of the New River and Pocahontas field and as high as 40 per cent in some other coals. The coal of the Connellsville district contains from 27 to 30 per cent.

In order to secure the advantage of coal the nearest possible to the exact composition desired, recourse is often had to mixtures of different varieties of coals. Mixtures of two are common and mixtures of three or more are sometimes used. Such mixtures enable the best results to be obtained in yield and quality of coke, in quantity of gas and reduction of coking time.

The advantage of making mixtures operates in the same direction as the value of the gas and the yield of coke in making it very desirable to place the ovens at the point of consumption of the coke rather than at the mouth of the mine. These considerations are so powerful that by-product ovens are almost always so located.

Coke as a Furnace Fuel

Coke has many of the most desirable characteristics for a blast-furnace fuel. Well-made coke from good coals is always in large lumps thereby supplying one of the fundamental requirements of a blast-furnace fuel. It has sufficient strength to resist breaking during the necessary handling which it undergoes both while it is being charged and during its descent through the furnace. It has a highly cellular structure which exposes the largest possible amount of surface per unit of weight. Consequently a very large amount of coke may be burned in a given area per unit of time, one of the necessities for rapid operation and high output of furnace.

The tarry constituents in the volatile matter of the original coal are completely destroyed by the coking operation, and there is, therefore, no tar to fill up the interstices in the charge and cut off the passage of the gases of combustion.

The worst objection to coke as a furnace fuel is its sulphur content. This varies from 0.6 per cent in the very best coke up to 2 per cent or more in inferior cokes. But when this limit is much excelled the fuel ceases to be suitable for the blast-furnace no matter how desirable it may be in other respects. In ordinary good cokes the sulphur will vary from 0.8 to 1.5 per cent.

The second objection is its content of ash derived from the original coal; this varies from about 6 per cent in extraordinarily pure cokes to 18 or 20 per cent in poorer ones. Ordinary furnace fuels range in ash from 8 to 12 per cent.

Phosphorus is ordinarily low, generally under 0.05 per cent, but its amount is important because it all goes into the iron and 0.03 per cent is the maximum amount allowed in some special grades of iron. As some phosphorus is always present in the ore and some in the flux the amount in the coke is of great importance in such cases.

The sulphur can be removed to a very large extent by proper use of fluxes, but the melting of the slag resulting from the use of these fluxes requires more coke and therefore increases the cost of iron made from high-sulphur fuel. Similarly the ash which generally consists in large parts of silica has to be fluxed and this is objectionable for the same reason.

It is, of course, obvious that the more ash the less fixed carbon and the fixed carbon is the only useful component of the coke. This ranges from 80 to 92 per cent, but good furnace cokes ordinarily contain from 85 to 90 per cent of fixed carbon.

Charcoal

It is an interesting fact that the development of the manufacture of charcoal has been along a similar line to that of coke there being three principal methods. Pits or meillers, beehived shaped ovens (ordinarily called kilns when used for charcoal), and retorts. There is no equivalent in the case of wood for the process in which distillation is performed for gas only and the resulting charcoal practically disregarded, as in the case of coal.

Charcoal is in general made from cord-wood, trunks and limbs of trees cut 4 feet long and in good practice when more than 10 inches in diameter, split down to a size equivalent to an 8-inch stick. There are two general classes of woods recognized in charcoal manufacture known as "hard" and "soft." The principal hard woods are hickory, hard maple, beech, yellow birch and several varieties of oak, while the soft woods embrace principally the resinous woods, very commonly the yellow pine and allied species.

The production of soft-wood charcoal is smaller and less important than that of hard-wood charcoal and the value of the coal produced therefrom is less for furnace purposes, it being lighter and softer.

Pits.—In the production of charcoal by this method a space of ground about 30 feet in diameter is leveled off and on this wood is stacked up on end in the form of a truncated cone two lengths or 8 feet high, leaving a chimney through the center. The entire structure is then covered with earth and leaves and fired at the bottom of the chimney. A supply of air is admitted insufficient for complete combustion. The volatile matter is distilled off and burns, supplying the heat to keep the operation going. This continues for a week or ten days, when the wood is all charred and the process completed except the cooling off of the charcoal by standing covered.

There is, of course, some loss of charcoal through excessive air supply and as a result the yield of charcoal per cord of wood in this process is poor, nor is there any recovery of by-products. As a consequence of these disadvantages this process has gone out of use almost completely.

Kilns.—These constitute the next step in the development of charcoal production. They are built of red brick because only low temperatures are needed for the charring of wood and

are very much larger than coke ovens, about 30 feet in diameter at the bottom tapering to about 24 feet at a height of 18 to 20 feet, the whole covered with a flat dome rising about 6 feet further. They are provided with three rows of draft holes at regular intervals, one immediately at the bottom, one about 3 feet higher, and the third about 3 feet above the second. An opening through one side of the dome, about 5 feet square, is provided through which they are charged, and a door about 5 feet wide and 6 feet high at the bottom through which the charcoal is removed, also an opening at the bottom on one side which communicates by a short chimney with a smoke main. They are also provided with a small hole about one foot in diameter in the center of the dome.

These kilns are filled with wood to the top, the wood being piled as uniformly and smoothly as possible with a stack up through the center. Fire is started at the base of the stack and the large top and side openings are then closed with thin steel plate doors and luted air-tight. The fire is permitted to burn very slowly with natural draft through the hole in the top of the dome for a few hours. This opening is then closed and the kiln is connected with the smoke main, on which suction is maintained by a system of fans. The draft is carefully regulated, the bottom row of holes being used first and the higher ones later in the operation. At the end of about 10 days most of the volatile matter is distilled off, then the kiln is disconnected from the main, the draft holes are sealed up airtight and the kiln is allowed to stand for a period of 10 to 20 days in order to cool the charcoal well below the ignition point.

Charcoal is extremely inflammable. It will ignite spontaneously even at atmospheric temperature under certain conditions; presumably, on account of its very great power of absorbing gases, the action would seem to be not unlike that of spongy platinum in igniting hydrogen.

The situation is further complicated by the fact that the use of water to extinguish fires in this fuel is highly objectionable. Charcoal absorbs vast quantities of water which appear not to be driven off except at a very high temperature so that the charcoal seems to descend almost to the hearth of the furnace still containing water and the effect of such water is exactly the same as if introduced at the tuyeres.

On this account charcoal must be cooled with great thoroughness before exposure to the air.

After the coal has been properly cooled all the doors are opened and the buggies which are to deliver the charcoal to the furnace are taken into the kiln and there loaded.

The shrinkage during the charring operation is very great, about 50 per cent. so that while the kilns are filled with wood they are only about half full of charcoal when opened.

The smoke drawn off through the mains by the fans as above described is subjected to cooling and scrubbing whereby its volatile constituents are removed and recovered, but in a very dilute condition because of the great quantity of water vapor in the smoke which comes from the water of the wood, because thoroughly seasoned wood contains about 50 per cent water. Of the other 50 per cent about one-half is charcoal and one-half volatile matter. Unseasoned wood contains much more water than this.

The valuable by-products in the case of wood are methyl alcohol, acetic acid, and wood tar. The latter is not so valuable industrially as coal tar and for that reason is commonly used only as a fuel to aid in the distillation process. The wood alcohol is obtained, first, in the form of a 2½ per cent solution. This is rectified by distillation up to 95 per cent or more. It is perfectly possible to make it chemically pure by proper distillation and a little chemical treatment.

The acetic acid cannot be recovered by distillation because its boiling point is too close to that of water and the two substances dissolve one another thoroughly in all proportions. The acetic acid is therefore recovered by neutralizing it with a mineral base, commonly lime, which forms a soluble but non-volatile acetate. This is recovered by evaporation of the water.

The Retort Charcoal Process.—This process is very similar to the by-product coking process, but on account of the lower temperature the retort is built of steel instead of fire brick. It consists of a steel chamber, roughly, 6 feet wide by 8 feet high and 50 feet long, surrounded with a brick setting and provided with a fire box under each end. A stack is provided to draw the products of combustion from the fire boxes around the retort. These retorts are charged with eight cords of wood on four-wheel trucks built of steel slats. The retort is closed at both ends by air-tight steel doors and is provided with an outlet for the volatile products of distillation.

After charging, the doors at both ends are closed and fires built in the fire-boxes. The gas outlet is connected to a water-cooled condenser and all the volatile products are condensed therein. The non-volatile gases are highly combustible and are led back under the retort to assist in supplying the heat it requires. In about 20 to 24 hours the process is complete, the doors are opened, the trucks containing the charcoal are withdrawn and immediately replaced by others containing a fresh charge of wood. The doors are then closed again and the process repeated.

The trucks containing the charcoal are drawn as rapidly as possible into the coolers which are steel chambers exactly like the retorts in general shape and size, but more lightly constructed and set out of doors so as to get the maximum cooling effect. There are two of these to each retort set in line with it and separated from it and from each other by a short open space. The charcoal is put first into the nearest one and there allowed to cool for twenty-four hours, and at the next drawing of the retort it is moved into the second one, the fresh charge being put into the first cooler.

On account of the rapidity with which charcoal burns on exposure to the air, the transfers, particularly the one from the retort to the first cooler when the charcoal is very hot, are made as quicly as possible and the air-tight doors of the coolers are immediately closed so that no air can reach the charcoal. Even with these precautions and with an additional cooling of 24 hours in open air, fires by spontaneous combustion of the coal are not infrequent.

This process gives much better yields of by-products than the kiln process. The latter in good practice yields four to five gallons of refined methyl alcohol and 85 to 100 pounds of acetate of lime per cord of wood. The retort process yields in good practice nine to ten gallons of refined alcohol and 170 to 220 pounds of acetate of lime.

The reason that this great superiority in the yield of valuable by-products has not forced the introduction of the retort more rapidly is that it suffers under two disadvantages. First, the cost of plant per cord of wood carbonized is about three times that of the kiln plant and for large capacity it runs into such a heavy expenditure as to put these plants out of the reach of any but relatively large and wealthy corporations. Second, the maintenance charges of the retorts and plant generally is very heavy, the bottoms are liable to be burned out by a few hours careless firing and the retort as a whole may be warped so as to be worthless without heavy repairs, unless it is managed with great skill.

Those who have installed and managed retort plants with skill and intelligent care think that they constitute the only method of wood distillation worthy of consideration, while those who have operated them carelessly and ignorantly declare with equal positiveness that from a commercial point of view they are inferior to kilns, in spite of the much lower yields of the latter, which are undoubtedly very much easier to manage.

The yield of charcoal in the two cases is not very different, about 50 bushels of 20 pounds to the bushel per cord of wood.

It is probable, however, that this comparison is not quite fair to the retort because it can and does drive off the volatile products more thoroughly than the kiln can do without waste of coal by combustion, so that while the weight of charcoal is about the same in two cases the fixed carbon in the retort coal is greater than that of the kiln coal.

This is partly compensated in the opinion of some operators

by the fact that the retort coal is more brittle and therefore breaks up worse in the furnace than the kiln coal. Such personal experience as I have had tends to contradict this view as I have found that better fuel economy was obtained with retort than with kiln coal.

In a large-scale charcoal operation the difficulty of obtaining an adequate and steady supply of cord wood is very great. The yield of cord wood per acre of timber land is from five to ten cords when only the material unsuitable for saw timber is taken and 40 or 50 cords per acre when all the timber is taken. The former condition is the more common and for this reason areas of several square miles must be cut over per year to supply a large charcoal operation. This makes many transportation difficulties.

In the earlier days of the industry small batteries of kilns were built close to the timber supply and the charcoal was shipped from these to the furnace, no attempt being made to recover the by-products. When commercial conditions forced the abandonement of this wasteful procedure the small batteries of kilns in the woods were abandoned and one large central battery was built preferably at the furnace, because the labor of handling the finished charcoal and its consequent breakage were reduced to a minimum, and, what is equally important, the waste gases of the furnace became available to supply steam for the great amount of distillation involved in the recovery of the by-products.

In those cases in which retorts are installed the same considerations hold and such plants are located at sites where timber may be shipped to them by rail from a large territory. Very frequently these also are located at the furnaces which consume their coal, but not necessarily because owing to the greater profit on their by-products they can sell their coal more cheaply than can the kiln plants and can, therefore, deliver it over considerable distances in competition with kilns.

Methyl alcohol is worth at the plant from 25 cents to about 40 cents per gallon, depending on the location and the perfection of the rectification.

Acetate of lime is worth from 2 to 2¾ cents per pound at the plant, depending upon the state of the market.

The cost of recovery with properly designed plants is low and therefore the income received from these by-products has a very important commercial bearing on charcoal production. In fact, the charcoal is the real by-product at many of these plants and the furnace is built simply as the most convenient means to consume this fuel.

Charcoal as a Furnace Fuel

Charcoal is almost an ideal furnace fuel. It is nearly free from sulphur, having only a few hundredths of 1 per cent as against approximately 1 per cent in coke and about 1 per cent of ash against about 10 per cent for coke. Moreover, this ash consists very largely of lime and alkalis, so that it supplies a part of the flux required for the gangue of the ore instead of being largely silica and requiring to be fluxed with additional bases as is the case with coke.

Charcoal has a highly porous structure and can therefore be burned at a rapid rate per square foot of hearth area. This fuel, however, is subject to one drawback. It has not quite the physical strength of coke and would perhaps be unable to stand the conditions existing in a large furnace. On the other hand, owing to the difficulties of wood supply above mentioned, charcoal furnaces are always small as compared with coke furnaces, and therefore the necessity of its having to withstand the pressures customary with large furnaces does not arise.

Charcoal has one bad characteristic not widely appreciated. In spite of its low ash (about 1 per cent) its percentage in fixed carbon is very low, only about 69 per cent, or less in kiln coal. The remainder, about 30 per cent, is volatile matter which is not driven off at any heat which it is permissible to use in charcoal distillation. This volatile matter is about one-half hydrogen and of the remainder a large portion consists of gases of high reducing power, methane and CO among the number.

Because of these valuable characteristics, the fuel per ton of iron produced is less with charcoal than with coke. These matters will be more fully discussed in a subsequent article.

Anthracite Coal

Anthracite coal has about the same chemical analysis as good coke for the reason that it is simply bituminous coal whose volatile matter has been distilled off under conditions of heat and pressure arising from the geological movements of the earth's crust. The pressure has closed up the cells as they were formed and anthracite instead of being cellular like coke is in fact the most close grained of any variety of coal. It only occurs in quantities of commercial importance in a small district in eastern Pennsylvania in the United States, though small deposits are known elsewhere, and deposits of much importance occur in Wales in Great Britain.

Owing to its abundance and cheapness in the early days of the industrial development of the United States it was the principal fuel used in the blast furnace until about 1860, when the development of the coking process began to furnish a rival fuel. As the magnetite mines of New Jersey and the limonite mines of eastern Pennsylvania are in close proximity to the anthracite coal fields, a great iron industry was developed based upon these materials.

But the increasing use of anthracite as a domestic fuel enhanced its value, while the development of the blast furnace proved coke to be the better fuel of the two for its purpose. The anthracite iron industry, therefore, declined, and there are only a few small furnaces using this fuel exclusively, though in the eastern Pennsylvania district, where it is cheaper than good coke, a certain amount of it continues to be used in admixture with coke to lower the cost of the fuel charge.

Anthracite as a Furnace Fuel

Anthracite has three disadvantages as a blast-furnace fuel when compared with coke.

First, it is entirely lacking in cellular structure and is simultaneously of much greater density, so that the ratio of surface exposed per unit of weight is only a small fraction of what it is in the case of coke and only a proportionately small amount can be burned per square foot of hearth area. This means that the output of a given furnace is necessarily much smaller when supplied with anthracite fuel than with coke.

Second, it has a tendency to spall or decrepitate under the action of heat. As a consequence of this spalling action the interstices through the fuel which form the principal passageway for the gases are greatly obstructed and the pressure required to drive the gases through the furnace is much increased. Moreover, this spalling seems to produce a tendency to scaffold the furnace, which will be further discussed in a later chapter.

Third, the much greater density of the fuel as compared with coke reduces the volume of the fuel charge very greatly. This increases the density of the charge as a whole and permits a longer time for the passage of the ore through a furnace of given size. But, for the reason already mentioned above this advantage is useless in increasing the output. On the other hand, the increased density of the charge means increased resistance to the blast.

As a consequence of these three conditions anthracite furnaces are characterized by slow driving and small outputs, high pressure, and a strong tendency to become scaffolded and work irregularly. These are the technical conditions, which, joined with the commercial considerations above outlined, have caused the decline of anthracite as a blast-furnace fuel.

Raw Coal

Certain coals, called "bituminous" for lack of a more descriptive name, while they contain a normal amount of volatile matter, are largely free from tarry ingredients. These are known as "dry" bituminous coals. On account of their relative freedom from tarry matter these fuels fall within the fundamental requirements of a blast-furnace fuel and in the past were quite extensively used as such.

But the development of the coke blast furnace has caused a retrogression in the use of this fuel very similar to that in the case of anthracite. In certain districts where these coals may be obtained cheaply and for the production of certain special irons they are still used in admixture with coke. But so far as known to me there are no furnaces now running in the United States on raw coal fuel exclusively.

Raw Coal as a Furnace Fuel

A lack of porosity and the consequent slow rate of combustion of these coals are presumably the principal reasons for the decline in the use of this fuel. Moreover, though these coals are relatively dry they are not entirely free from tarry matter, and as but a small quantity of tar is required to obstruct the passage of the gas, this also has tended to reduce the rate of driving possible with this fuel. Moreover, the tendency for the furnace to scaffold and work irregularly even with a small amount of tarry matter present in these coals is pronounced. Their use is, therefore, generally confined to a district in southern Ohio, where they occur and are applied in admixture with coke to the production of an iron containing 15 to 20 per cent of silicon and known as ferrosilicon.

The characteristics of these fuels have modified furnace practice, and this in turn has modified equipment and product, so that the classification of iron by the fuels used in their production is fully justified. But as will be seen from what has already been said the only units in the classification which have preserved their individuality unchanged are coke and charcoal, the only two varieties of iron now in common commercial use.

Additional Requirements

We have given a definition of the fundamental requirements of the blast furnace. To these may be added two more, which are necessities in the case of more than 99 per cent of the iron produced.

First, the slag must be sufficiently fusible to run freely from the hearth at a temperature easily attainable.

Second, the blast must be warmed before being delivered to the furnace.

The single exception to these two conditions occurs in the case of furnaces making cold-blast charcoal iron of which probably not over twenty or twenty-five thousand tons per year are produced out of a total of some thirty-five million tons in the United States. In this case the temperature attainable is very low and in consequence the slag is not heated hot enough to run freely and has to be removed from the furnace with the assistance of mechanical means such as slag-hooks, etc., or, if it comes from the furnace without assistance, it chills before going more than a few feet and is accordingly troublesome to remove.

Leaving out of account this very small percentage, all blast furnaces are blown with blast heated from 500 deg. to 1400 deg. F. and their slag is heated to a degree sufficient to enable it to run from the furnace without assistance and far enough to reach some convenient means of ultimate disposal. The heated blast was introduced by the Englishman, Neilson, about 100 years ago, and it has made possible the modern development of the blast-furnace because the velocity of combustion of the fuel is so greatly augmented by preheating the blast, and the continuity of the operation has so increased by raising the molten iron and slag well above their fusion points, that outputs have been increased almost a hundred fold since.

The iron blast furnace is now probably the largest industrial apparatus in use, not only in size of plant, but in the quantity of raw materials consumed and output. Furnaces have often been built which produced 600 tons of iron per day, but modern practice tends to slightly smaller units, furnaces now being built only for an output of about 500 tons per day; but this is maintained as an average output for months and years together.

To produce a ton of iron under favorable conditions requires about 2 tons of ore, ½ ton of flux, 1 ton of fuel and about 4 tons of blast; greater quantities are required when the conditions are less favorable. With each ton of iron is also produced about 6 tons of waste gases and about ½ ton of slag, this, as well as the iron, being handled in the liquid condition. For a 500-ton furnace we have, therefore, to handle every twenty-four hours about 1700 tons of solids, 750 tons of white-hot liquid and 5000 tons of hot gases (blast and waste gases together).

To do this on an industrial scale and with a labor cost amounting in good practice to less than 75 cents per ton, a great plant is necessary. Such a plant must contain the following parts: First: The shaft furnace itself, through which the entire mass of material just mentioned passes, the solids passing down to the liquid condition at the bottom and the gases upward through them. Second: Means for handling the raw materials, ore, fuel, flux, proportioning them properly and delivering them in certain accurate and positive relations into the top of the furnace. Third: Means for raising the atmospheric air to a pressure sufficient to force it through the furnace in the quantity desired. Fourth: Means for heating this supply of air uniformly and regularly to the temperature desired. Fifth: Means for handling the molten iron on its discharge from the furnace and turning it into a commercially salable product. Sixth: Means for removing the slag from the vicinity of the furnace. Seventh: Means for removing the gases from the top of the furnace and (as these gases are a valuable fuel) for leading them to those portions of the plant where they can supply the energy for compressing the blast and heating it. There is an eighth requirement not embodied in our fundamental definition, but which has become an absolute necessity notwithstanding; a supply of water for cooling certain parts of the furnace and preventing the intense heat inside from destroying the structure.

Available Alkali in Lime

The **South African Engineering Standards Committee,** supported by six South African scientific and engineering societies, has just published Pamphlet No. 3 of its Chemical Sub-committee, entitled: "Lime, Standardization of Methods of Sampling and Determination of Available Alkalinity." (Price 1s. net, South African Engineering Standards Committee, P. O. Box 1176, Johannesburg, S. Africa.)

The pamphlet deals with the uses of lime, method of manufacture, transport, and handling, describes methods of sampling unslaked lump lime, crushed lime, and slaked lime, and finally gives the following standard method for the determinations of "available alkalinity" (CaO).

"The sample as delivered for analysis is contained in an air-tight vessel, having been passed through a 30-mesh sieve. It is crushed with a Wedgwood mortar and pestle and the whole passed through a 60-mesh sieve, this operation being performed as quickly as possible. It is then placed in a clean, dry, wide-mouthed bottle fitted with a tight-fitting dry glass stopper so as to prevent any access of air. Two grams of this are carefully weighed out and agitated with 1 liter of a 2 per cent cane sugar solution (or 1 gram with ½ liter of 2 per cent sugar solution). If a shaking machine be available, 2 hours' continuous agitation should be given, if not, 6 hours' intermittent agitation, every care being taken to prevent coagulation of the lime, in order that the lime and solution may be brought into the most intimate contact during this period. When the agitation is finished the solution is filtered as quickly as possible and aliquot portions titrated with N/10 or N/5 acid, using rosolic acid as indicator, avoiding delay so as to obviate undue exposure to the atmosphere.

"The distilled water used in the above determination must be made neutral to rosolic acid to counteract the presence of dissolved CO_2."

The metal production of Alaska for 1912 was valued at more than $22,000,000, divided approximately as follows: Gold, $17,000,000; silver, $317,000; copper, $4,800,000. The development of the coal still awaits a settled policy on the part of the government.

The Radium Institute St. Joachimsthal

By Joh. Härdén

The medical treatment of diseases by means of radium and radioactive substances has now gained such importance that the great general interest shown both by laymen and scientists seems quite justified. It will be remembered that the Curie's discovered the new element radium in a mineral called pitchblende, which they obtained from St. Joachimsthal about 18 years ago.

During a recent visit to Bohemia the author took the opportunity to visit the quaint old town in the Bohemian mountains, and in the following lines a short report of its important Radium Institute and Factory will be given.

St. Joachimsthal is an ancient township, situated some 20 miles to the northwest of Carlsbad; it was once famous for its silver-bearing mines and smelting works, records of which are in existence as far back as 1515, when the town counted "over 1200 houses" and the smelting works are doubtless still older. By the way, it may be of interest to record, that it is assumed that the name "Thaler" the former German coin-unit has its origin from the silver gained in St. Joachimsthal.

As the mines were exhausted as regards the silver during the following 300 years, the importance of the town as a mining and smelting center diminished; also the war times proved disastrous to the industry, and to-day there are scarcely even ruins of the old silver smelting works left, while the population has dwindled down to only a couple of thousand, barely thriving on lace making, paper box manufacturing, and some revival of mining.

As the yield of silver diminished, the leading men naturally turned their attention to other sources of income from the minerals available. This part of Bohemia is remarkable for its great variety in various minerals; so, for instance, at Schlaggenwald, in about equal distance southwest from Carlsbad, the author found a tin and tungsten mine in full swing on the remainders of an old tin smelting industry and on the old dump heaps a veritable treasure for the geological student was in open daylight. The chemist stated that no less than 95 different minerals, more or less interesting, could be gathered here. Indeed, a large railway viaduct, built from the material taken from the old dump heap, showed a curious admixture of colors, copper pyrites, turmalin, small amethyst and garnet crystals appeared on the fracture of the hewn stone.

In the year 1853, the chemist Patera, appointed by the Government to investigate the possibilities of St. Joachimsthal, succeeded in finding a practical method for the manufacture of uranium dyes, an industry which proved very remunerative; for instance, in 1886 the factory sold over 11,000 kilograms of these dyes for about half a million marks.

The element uranium was, according to records, discovered by Klapproth in the year 1789 and the metal itself was isolated by Peligot, 1840, both from the uran-pitchblende from St. Joachimsthal. This mineral consists in the main of uranium oxyduloxyd, together with quartz, iron, cobalt, lead, bismuth, arsenic, calcium, magnesium, copper, vanadium, selenium, and in particular the radioactive substances "radium," polonium and actinium.

The discovery of these radioactive elements by Mr. and Mrs. Curie, in the laboratory of Professor Bequerel in Paris in 1896 is too well known to be further described here. The author had the privilege of being present when the late Professor Curie held his lecture on his far-reaching discovery, assisted by his skilled wife, Madame Curie, on the occasion of their receiving in joint the Nobel Prize in Stockholm in 1905, and it is safe to say that the unostentatious lecture, accompanied by a number of startling experiments, showing the apparent miracle of the birth of a new element, made a lasting impression upon those present.

Mr. and Mrs. Curie had, upon request of Professor Bequerel, commenced the investigation of radioactivity of uranium salts derived from Joachimsthal, and as they soon found that a new and unknown element was present, they obtained a larger quantity of the slimes remaining from the manufacture of these salts. In this they found a more considerable source of the costly substances, and this slime is even to-day the chief raw material for the recovery of the radium.

The isolation of the new element is rather simple in principle, being a process of dissolving and repeated fractional crystallization, until, at least in Joachimsthal, the pure radium carbonate remains, which is subsequently transformed into bromide or chloride as the demand calls for. But the practical side of the problem is less simple, inasmuch as the element is so rare, that several tons of radium-bearing slimes must be treated in order to extract only a few milligrams, and this, of course, makes the process rather costly.

In addition to the radium salts are also the radioactive waters to be counted. These waters contain the so-called emanation, which is assumed to be a disintegration substance of radium and as such is found to be highly radioactive.

A unit for the radioactivity has been agreed on by the scientists, similar to the units for electric potential, the volt, and the ampere for the current. The unit for radioactivity is called "Mache" after Prof. Heinrich Mache in Vienna, who together with S. Meyer carried out important work on this subject. They soon found that some kind of unit was necessary, and recommended the use of the units of the "Absolute" or the "C. G. S." system. This unit is represented by the strength of the saturation current, which is caused to pass through by the emanation arising from one liter of the water to be tested; it is expressed in electrostatic units. As this, however, would give a too low figure for practical use, it is multiplied by 1000 or 10^3, and it is this quantity which is generally called "One Mache."

To give an idea of the strength of radium emanation in various springs, counted in Mache units per liter of water, the following table may serve as a comparison:

St. Joachimsthal	600 units.
Gastein	54 to 155 units.
Baden-Baden	24 to 126 "
Bath (uncertain)	20 to 86 "
Carlsbad	1.4 to 50.2 "
Muenster St.	13.5 to 23.4 "
Teplitz	3 to 8.7 "

These waters are either used internally, or chiefly, as baths, according to the prescription of the doctors. It is, of course, outside the scope of this report to enter into a description of the various cases of which these curative waters of high strength are prescribed; may it be sufficient to mention that very excellent results have been obtained in cases of nervous breakdowns, ischiatic, etc., while the direct application of radium, as well known, is rendering important relief in cases of cancer, ulcers, etc.

The active waters at St. Joachimsthal are conducted direct from the mine, at great depths, into the curehouse, so as to render their curative action as soon as possible to the place of application.

The waters may also be stored and sent to distant places in suitable vessels; it should be borne in mind, however, that the emanation is gradually giving out when so stored, the waters losing their strength at the rate of one-half in 3¾ days and this goes on regularly, so that a water containing originally 600 units, has only a strength of 300 after 3¾ days, 150 units after 7½ days, 75 units after about 11 days and so forth.

The emanation substance itself is a material of greatest interest physically: it appears in form of a gas which may be isolated and weighed; it is soluble in water, and is capable of passing through porous and gaseous substances. It is also constantly developing energy in form of heat, in fact, one kilogram (2.2 lbs.) of emanation is capable of developing 20,000 horse powers in one hour, if the whole of its energy could be spent accordingly. It may be condensed to a liquid at low temperatures, about 150° C, which shows all the characteristics of the gas. The spectrum is very distinct, though the gas itself is not visible, and non-inflammable. Boiling drives it out of its solution, also shaking, or if a current of air is driven through.

Glass and gems are slightly colored by the action of the emanation, water is slowly decomposed into hydrogen and oxygen, photograph plates are darkened; if strongly concentrated, it is observed to glow in the dark and to render other substances phosphorescent, it also ionizes the air, so as to conduct electricity. It is inert against practically all chemical reactions.

The emanation was discovered through its property of sending off "alpha" rays, the first range of rays given off by radium itself. It seems that the emanation is an instable substance, which as it is emanating the alpha rays, is disintegrating into a substance of lower atomic weight. The precipitate, which is accumulating with somewhat greater velocity on bodies charged withwith negative electricity, is again an instable substance, which is in its turn deteriorating, thereby giving off both alpha, beta and gamma rays. A product of this substance, called radium D has been specially studied by Dr. Gudzent and found to be of great importance for the medical treatment of some diseases.

Apart from this, the emanation is also splitting up into the gas helium, a well-known element, and this seems to be the lowest step of disintegration.

This gas was discovered by Jansen and Lokyer (according to Rutherford) in the sun's spectrum, in the year 1868. Nearly 30 years passed before this gas was identified as an earth element, and then it was only found occluded in uranium-bearing minerals, a coincidence which was rather puzzling, as the connection between uranium, radium and helium was at that time unknown; to-day, however, we understand quite well that the disintegration of the radium present in these uranium and thorium minerals had most likely been the source of the occluded helium. R. J. Moos found, when pitchblende residues from St. Joachimsthal were pulverized and treated in vacuum at high temperature, that 1.17% of helium could be extracted.

The emanation, however, is found in smaller or greater quantities, practically all over the earth, since radium is present almost everywhere, though in minimal quantities. But only a few places can boast of such riches in radioactive materials as St. Joachimsthal; the only other source of importance in Europe is the Trenwith mine at St. Ives, Cornwall, where considerable quantities of radium are mined daily, for subsequent purification for the Radium Institute in London.

Reports are reaching us from time to time claiming discoveries of new deposits both in the United States and in Australia; but to the author's knowledge, the bulk of the world's supply of this valuable agent for fighting disease is still prepared from the minerals from St. Ives and St. Joachimsthal. The daily papers are just now mentioning a project, according to which the rich mines at St. Joachimsthal are to be bought up by an English concern, for the stated sum of £700,000. This price may appear high, at a first glance, but if we consider the extraordinarily high price of the radium, amounting to many pounds sterling per milligram (the prices fluctuating with the market) caused by the rarity of the substance and the tedious labor of extracting it from the ores, one wagonload of crude ore only yielding a few milligrams of radium, it is easier to understand why the price should be so high.

The radium factory at St. Joachimsthal is only a small one, compared with the world's demand, but it is no doubt capable of considerable developments, similar to those of the British Radium Institute; given time and means of carrying out all modern improvements.

Luton, England. JOH. HÄRDÉN.

The Engineers' Society of Western Pennsylvania has organized a metallurgical and mining section, the membership including chiefly the metallurgists of steel mills and allied industries in the Pittsburgh district and mining engineers throughout the Western Pennsylvania, West Virginia, and Ohio coal mining districts. At the first meeting of the section Mr. B. G. Klugh lectured on sintering processes for iron-bearing materials. Besides the metallurgical and mining section, the Society has a mechanical and a structural section.

The Art of Enameling or the Coating of Steel and Iron with Glass*

By Raymond F. Nailler

Before considering the technology of any particular type of enamel it will be well to consider the meaning conveyed by the word enamel. If the present-day enamel be briefly and scientifically designated as a boro-sodium-potassium-aluminium silicate generally colored by metallic oxides, then the following definition given by Popelin deserves to be quoted since this, notwithstanding its length, may be described as most clear and comprehensive: "Enamel is a glass fusible at a low temperature and usually compounded of borates and silicates. This mixture, originally colorless, combines with the greatest ease with metallic oxides under the influence of a pyrotechnic operation, thereby acquiring various colors according to the nature of the oxide which the enameler can vary at will."

The uses to which the general class of ceramic compounds known as enamel is put are varied. We have noted that enamels are used for artistic or decorative purposes. Parallel to this use we may classify a far more important type of enamels as "commercial," and it is with this type of enamel that we are to concern ourselves.

The first class of commercial enamels is that used on cast iron, and this classification includes the field of sanitary equipment, together with various forms of cast-iron kettles and similar pieces of engineering apparatus.

The enameling of sheet steel may be taken as the second class of commercial enamels. The most familiar division of this class includes cooking-ware enamels. The various forms of so-called agate and granite ware as well as the single colors are made by enameling sheet-steel forms.

The second class of steel enameling which has more recently come into industrial importance is the manufacture of enamel signs, and the third field includes the manufacture of heavy equipment for large-scale food preparation, the dairy industries and general chemical operations. The operations in this last case may take the form of tanks, kettles, evaporators, pipes, etc.

In outlining the technology of the enameling industries as a whole we may include to a certain extent all of the above commercial classes, certain details of which are varied to meet the conditions of cast-iron and various types of sheet-steel products. The outline following, however, is characteristic of all classes.

The first step to be considered is the preparation of the enamel. The purity of the raw materials to be used in the compounding of an enamel must be certain, and in the cases where materials can vary in strength the actual analysis of the substance must be known to assure proper results. In many cases the secret of an enamel lies in its formula and so the compounding of the batches is very carefully guarded, only a person of responsibility having charge of weighing the ingredients. The various materials are generally kept in bins which are numbered, the person in charge of the mixing having the formula stated in terms of these numbers. The properly weighed batch is thoroughly mixed, this being accomplished either by shoveling carefully on a specially prepared floor or by mechanical mixing by means of a rotating agitator.

The product of the mixing room is taken to the smelter in which the various ingredients are fused together in the form of a mass having the characteristics of glass. The furnace in which this operation takes place is a special reverberatory furnace similar to that used in the puddling process in the manufacture of wrought iron. The coal is placed on a grate at the front end of the furnace and the burning gases pass over the bridge wall, strike the roof, and are deflected against the batch. The first change which may be noted in the smelter is the driving off of the water from the borax which produces a swelling up of the mixture. The fluxing materials then melt down and as the heating is continued, dissolve the

*Extracts from a paper published in the October issue of the *Journal* of the American Society of Mechanical Engineers. With the exception of the historical introduction the paper is here given practically in full.

more refractory constituents. If the batch is properly stirred and the temperature of the furnace carefully regulated, the product of this operation is a clear transparent glass containing no particles of unfused material. When a test of the fusion comes up to this standard the furnace is tapped and as the liquid glass flows into a tank of cold water it is broken up by the chilling action of the water, the result being a thorough granulation of the product. The material thus prepared is known as "frit."

The next step in the preparation of the enamel involves grinding the frit to a fineness at which it can be applied to the surface of the metal. The mill here used is the ordinary pebble mill lined wth porcelain brick and containing very hard flint pebbles. In the cast-iron industry two general processes are used, the dry and the wet, in sheet-steel work only the wet. In the first case the frit is ground dry, the pulverized enamel being sprinkled on the hot cast-iron piece. In the wet process a certain percentage of pure white clay is placed in the mill with the frit and a certain definite amount of distilled water. When certain grades of enamel are to be manufactured various compounds are also added here to aid in producing desired colors, gloss and opacity. An instance of this is the addition of tin oxide in the mill in the production of white cooking utensil enamels. The fineness to which the enamel is ground depends upon the particular use to which it is to be put, manner of application and other factors. In the wet process the essential feature is that it be fine enough to remain suspended in the water assisted by the clay and the so-called "vehicles" mentioned below.

At this point we may well take up the consideration of the construction and preparation of the material to which the enamel is to be applied. Cast-iron enamels are applied to castings of the desired shape. In the case of cooking ware enamels the shapes are constructed by pressing and spinning. For heavier equipment such as jacketed kettles the apparatus is constructed by riveting or, preferably, welding the sheet steel in the desired form. The selection of the steel with a view to its chemical analysis is of prime importance. A reliable specification reads as follows:

Sulphur	below	0.040
Phosphorous	below	0.030
Manganese	about	0.40
Silicon	about	0.010
Carbon	about	0.10

The steel must necessarily be free from laminations or other mechanical imperfections.

With reference to the construction of heavy apparatus the sheet ranges from 3/16 inch to 3/8 inch in thickness. Two general methods of construction are in use: The first involves the formation of unit sections fitted with flanges. These sections are enameled separately, bolted together at the flanges and the desired apparatus so constructed. The preferable practice, however, is to construct the apparatus in one piece by means of autogenous welding, thus avoiding the use of gaskets or other packing materials in erection.

That the enamel may properly adhere to the surface of the metal the latter must be free from dirt or scale, and in the case of welded joints the welds must have a preliminary grinding to reduce the roughness. The entire surface of the apparatus is then cleaned by pickling or sand blasting, the latter process being altogether used in cleaning large apparatus. As the crude ware leaves the sand blast it has a roughened, clean, metallic surface and is in the proper condition to receive the enamel.

Before applying the enamel to the metallic surface it is prepared by a process known as "setting up." This involves the addition of certain chemicals to the enamel as taken from the mill, the function of this addition being to assist the clay in holding the enamel particles in suspension. Substances so added are termed "vehicles." At this state the enamel must be diluted with distilled water to the proper consistency for application.

In applying the enamel to the metallic surface three general methods are in use: The first, applicable to small pieces only, is known as "dipping," the piece being dipped into the enamel, the excess of which is shaken off, leaving a thin coating on the metal. The second method, known as "slushing," involves pouring the prepared enamel over the surface and allowing it to drain. The third method, which is the principal one used on larger apparatus, involves spraying the finely ground enamel on the metallic surface by means of the compressed air atomizer.

Preliminary to the consideration of firing the enamel we may well review the types of furnaces in use. The first type is known as the muffle furnace and involves the use of a large fire clay oven externally heated by means of coal, gas or other fuel. The apparatus to be fired is placed on suitable supports in this muffle. The other type of furnace is known as the direct fire furnace, in which the heat from the fire is taken up by the walls of the firing chamber and radiated to the apparatus placed within the chamber on suitable racks. This general class of furnace has two special divisions in that, on the one hand, the piece is rotated within the furnace, while on the other hand, the piece is allowed to remain stationary. For small work the muffle is in general use, but for the production of large apparatus the direct-fire furnace is necessary. At first thought, the muffle furnace may be considered to have an advantage in that the products of combustion together with the dust from the fire cannot come in contact with the enamel. Again it may seem that a more even heat can be realized in the muffle. On the other hand, with a properly designed direct-fire furnace in which the combustion is complete before the gases reach the firing chamber no trouble is experienced, due to their presence or to the dust from the fire. The use of natural gas further does away with this latter possibility.

In a furnace for firing smaller ware the charging mechanism is a fairly simple matter, it being necessary merely to place the material in the furnace by means of a small fork operated by hand or mechanically. But in the manfuacture of engineering apparatus where a single piece may weigh 3000 to 4000 pounds, it is necessary to have a large mechanical charging machine on which the piece may be placed outside the furnace, the arm of the machine then properly placing it in the furnace. The general design of such a machine suggests the charger used in open-hearth practice. The apparatus to which the enamel has been properly applied is placed in the furnace, which is maintained at the proper temperature. This temperature varies with the nature of the enamel and in cases of high-silicon acid-proof enamels reaches in the neighborhood of 2500 deg. F. The control of the burning is made possible by the changes which occur in appearance of the enameled surface as fusion takes place. At first the fine particles of enamel begin to fuse together and a general blister condition exists, giving the surface a very dull appearance. But as the enamel matures this dull appearance gives way to a bright glass, which, when properly developed over the inside surface, is indication that the piece should be withdrawn from the furnace. The time required to burn a piece properly depends upon the temperature of furnace, thickness of metal, and nature of enamel.

Nothing has been said so far as to the composition of the enamel or of the number of coats applied. In general there are two kinds of enamel, know as ground coats and cover coats. The former serves as a bond between the enamel and the steel, and the latter serves to build up the body of the enamel and presents the finished surface.

In the ground coat color is no object. Its composition is such as to render it adherent and strong. In addition to the ordinary components cobalt oxide seems to be essential to the production of adherence. The explanation of this is debatable.

The cover coat is the one which forms the major part of the enamel and if definite color, opacity, etc., are objects, the necessary ingredients for their production are introduced here, assisted by mill additions as noted above.

In the manufacture of acid-proof enamel the cover coat is essentially a high silicate and must be free from any metallic oxides, such as oxide of tin, lead, iron, etc.

The piece to be enameled receives one ground coat which is burnt well into the steel at a high temperature. The cover coats may be one or two in number for ordinary enameling, but should be at least triple for acid-proof work. In the production of acid-proof apparatus the use of a ground coat or cover coat is now eliminated and the same material is employed for both which in place of being an enamel, as commonly termed, is in reality a boro-silicon glass and without the use of any metallic oxides in its compounding.

It is very interesting to note the chemical changes which take place in the various stages of the production of the finished enamel. Avoiding as far as possible deep technicality, they may be summed up as below. As a starting point, let us select a cover coat formula used in the production of a "dark blue" cooking ware enamel.

Feldspar, lb.	120	Saltpeter, lb.	7
Quartz, lb.	72	Oxide of Cobalt, lb.	7½
Borax, lb.	80	Oxide of Manganese, lb.	1
Cryolite, lb.	30	Clay in the mill, per cent.	4

First considering the behavior of each of the constituents when heated we can later clearly note the general reactions which take place when they are fused together in the smelter.

Feldspar varies greatly in composition but as commonly used in enamels is an aluminium-sodium-potassium silicate of approximately the following analysis:

	Per Cent
Silica (SiO_2)	70
Alumina (Al_2O_3)	17
Soda (Na_2O)	7
Potash (K_2O)	6

On smelting, none of these constituents vaporize, so we may consider them all later in the smelting of the above enamel.

Quartz introduced as a pure glass sand contains practically 100 per cent silica (SiO_2).

Borax is chemically known as sodium tetra-borate, and in the crystalline form as used has a certain definite amount of so-called water of crystallization which must be taken into account. The formula $Na_2B_4O_7.10H_2O$ is resolved into $Na_2O.2B_2O_3.10H_2O$. Without going through the calculations assume this to correspond approximately to the following analysis:

	Per Cent
Water (H_2O)	16.0
Boric Oxide (B_2O_3)	37.0
Soda (Na_2O)	47.0

Of these the last two do not vaporize on smelting, but the water is evaporated, hence 100 lb. of borax smelts to 84 lb. of the remaining oxides.

Cryolite is a double fluoride of sodium and aluminium, the formula of which may be written Na_3AlF_6. When smelted the sodium and aluminium appear as oxides and from 100 lb. of cryolite we realize about 24 lb. of alumina (Al_2O_3), 44 lb. of soda (Na_2O) and 54 lb. of fluorine (F_2). There is some dispute as to whether or not the fluorine is vaporized. In the present discussion it matters not, hence we shall consider that the third of these three compounds is lost in smelting.

Saltpeter is potassium nitrate KNO_3. When heated under the conditions of smelting it may be considered to break up into potash (K_2O) and nitrogen pentoxide (N_2O_5). The reaction is $2KNO_3 = K_2O + N_2O_5$. By calculations based on this reaction 100 lb. of saltpeter yields about 47 lb. of potash (K_2O) and 53 lb. of nitrogen oxide (N_2O_5). The latter may be considered as completely vaporized.

Oxide of cobalt (CoO) may be taken as non-volatile and pure. The same may be assumed for the oxide of manganese (MnO_2).

To sum up the above as an outline of the reactions taking place in the smelter

120 lb. Feldspar	gives	$120 \times 0.70 = 84.0$ lb. Silica (SiO_2)
		$120 \times 0.17 = 20.4$ lb. Alumina (Al_2O_3)
		$120 \times 0.07 = 8.4$ lb. Soda (Na_2O)
		$120 \times 0.06 = 7.2$ lb. Potash (K_2O)
72 lb. Quartz	gives	$72 \times 1.00 = 72.0$ lb. Silica (SiO_2)
80 lb. Borax	gives	$80 \times 0.16 = 12.8$ lb. Water (H_2O) (Vaporized)
		$80 \times 0.37 = 29.6$ lb. Boric Oxide (B_2O_3)
		$80 \times 0.47 = 37.6$ lb. Soda (Na_2O)
30 lb. Cryolite	gives	$30 \times 0.44 = 13.2$ lb. Soda (Na_2O)
		$30 \times 0.24 = 7.2$ lb. Alumina (Al_2O_3)
		$30 \times 0.54 = 16.2$ lb. Fluorine (F_2) (Vaporized)
7 lb. Saltpeter	gives	$7 \times 0.47 = 3.3$ lb. Potash (K_2O)
		$7 \times 0.53 = 3.7$ lb. Nitrogen Oxide (N_2O_5) (Vaporized)
7½ lb. Cobalt Oxide	gives	$7\frac{1}{2} \times 1.00 = 7\frac{1}{2}$ lb. Cobalt Oxide (CoO)
1 lb. Manganese Oxide	gives	$1 \times 1.00 = 1$ lb. Manganese Oxide (MnO_2)

We shall consider that of the above the H_2O, N_2O_5 and F_2 are vaporized. This leaves for the constituents of the frit (totals of the above)

	Lb.	Per Cent
Silica (SiO_2)	156.0	53.5
Alumina (Al_2O_3)	27.6	9.5
Soda (Na_2O)	59.2	20.3
Potash (K_2O)	10.5	3.6
Boric Oxide (B_2O_3)	29.6	10.2
Cobalt Oxide (CoO)	7.5	2.6
Manganese Oxide (MnO_2)	1.0	0.3
Total	291.4	

The loss, theoretically, on smelting is plainly $317.5 - 291.4 = 26.1$ lb. or $\frac{26.1}{317.5} = 8.2$ per cent. (317.5 lb. = original weight of batch).

The mill additions together with the agents used in "setting up" are fused with the frit as the piece is fired. By a process similar to the above we could compute the final composition of the enamel by taking these additions into consideration. The actual amount of material added in this case, however, is so slight and of such a nature that the change resulting therefrom is not sufficient to affect materially the composition of the enamel. When an addition of about 12 per cent tin oxide accompanies the clay a very significant change in the composition of the enamel is produced.

It may be well to note in passing some of the means by which various colors are produced in the enameling industries. It will be impossible to enter into great detail without taking too much time, but the mention of certain compounds in connection with the colors produced by their use will serve our purpose.

The production of a good white enamel either for casting or sheet-steel work may be said to depend, at the present time, upon the use of tin oxide. Great have been the efforts to substitute less expensive substances, such as compounds of antimony and lead. But an antimony white which looks good alone is plainly seen to be off-color when compared with a good tin oxide white.

Going to the other extreme of color, black, we encounter difficulties. There is any number of formulæ for black enamels, but when the results are closely compared we find that the colors range widely through brown blacks, blue blacks, purple blacks, etc. Certain compounds of manganese and iron used together give a color approaching black. Other formulæ call for the combined use of oxides of manganese, cobalt and copper. Again we find oxide of nickel added to the above three oxides.

A color much seen in enamels is blue and the use of cobalt is very satisfactory in the production of this color in various

grades of intensity. Manganese alone produces purples and violets, and in combination with cobalt gives various shades of purple-blue.

Green enamels are chiefly produced by the use of chromium oxide and copper oxide, while in some cases a mixture of copper and cobalt oxide is used.

Reds of various shades are produced by the use of red oxide of iron. In connection with it we find that tin oxide aids greatly in giving capacity and bringing out the color. In the production of brown enamels we may use ferrous chromate. Various yellows are produced by salts of cadmium, chromium, and uranium.

The more delicate shades of rose and purple are produced by the use of gold compounds. So-called "pink rose" is used in the manufacture of certain artistic enamels. Perhaps the best known gold compound used in enamel coloring is "purple of Casius." The exact composition of this product is a question. It is made by the combined use of auric, stannous, and stannic chlorides. The color produced is also commonly called "purple of Cassius."

Before drawing this paper to a close, attention is invited to a general consideration of the future of the enameling industry. Neglecting art enameling and sign making we come to the field of steel enamels. So far as the cooking ware industry is concerned, the field is practically constant. Granting that the demand for that class of article is increasing, as the public becomes accustomed and educated to its use, there is an opposing tendency in the rapidly increasing use of aluminium ware. Exactly how these and other factors now balance would be difficult to ascertain. But aluminium is a metal and its metallic properties cannot be denied. Under certain conditions it is attacked by various substances used in the culinary arts and a contamination of the preparation is inevitable. No doubt the time will come when a high silica enamel known to be free from tin and other poisonous compounds will enter the cooking ware field. The government is becoming more and more careful in protecting the public from foods of injurious nature, and it is not too much to expect that soon it will establish more rigid restrictions relative to the ingredients entering into the manufacture of apparatus in which food is to be prepared. At such a time an enamel coming up to requirement will be free from injurious compounds and will come into great demand.

In the preparation of foods on a factory scale we find an enormous and constantly increasing demand for larger pieces of enameled steel apparatus in the form of pans kettles, tanks, pipe, etc. There are many lines of pressure being brought to bear both by the government and public opinion which lead to the conslusion that the increasing demand for this style of apparatus is without limit. Canning and preserving factories and dairy establishments have found a large use for copper and tin in the construction of containers, vacuum pans, etc. The acids of fruit juices and vegetable pulps have a very marked action on these metals and the resulting contamination of the product is known to be of danger to the consumer. The use of an enamel containing tin, lead or other metallic oxides is but the first step in the right direction. The presence of these metallic oxides in the enamel renders it corrodible and contamination results. The solution is the use of an acid-proof enamel free from all such poisonous substances. In the milk industry a similar line of reasoning applies. Further compare the ease of maintaining sanitary conditions in a one-piece enamel-lined unit with the trouble experienced in the use of a metal container or even an enameled article made up of composite parts between which are gaskets.

Another consideration relates to the preparation of chemicals later used in food products, for instance, baking powder. Many operations connected with the manufacture of such products have been carried on in lead or other metallic pans and the resulting contamination has given no end of trouble. Acid-proof enamel is rapidly solving this problem also.

Finally consider the chemical manufacturing processes now carried on in apparatus of lead, wood and earthenware, necessitated by the corrosive actions of the liquors and gases involved. This includes the pharmaceutical field which alone is a matter of great importance. In all these and many other lines the use of acid-proof enameled apparatus is rapidly finding and filling a great demand.

Not only does steel apparatus meet the demands of modern industries, but in case a cheaper product is desired and at the same time a heavier construction is permissible, acid-proof cast-iron apparatus has its field. The possibility for size and variety of construction is, of course, more limited than in the case of sheet-steel apparatus.

In view of these considerations and many others which these have called to mind, we cannot but conclude that the use of enameled apparatus has just begun and with this extension of the long known art of metal enameling, a field of great industrial possibilities both for manufacturer and user has been opened. We may not be criticised as being over optimistic when we predict that in their ultimate stage of development the enamel industries will be ranked among the greatest of commercial activities. At such a degree of development the enameling industry will in no way deserve classification among the lost arts.

The Glucose and Starch Industry in the United States

Final statistics of the manufacture of glucose and starch in the United States for 1909 are given in detail in a bulletin soon to be issued by Director Harris of the Bureau of the Census, Department of Commerce. It was prepared under the direction of **W. M. Steuart,** chief statistician for manufactures.

Of the 118 establishments canvassed in 1909, 93.2 per cent were engaged primarily in the manufacture of starch, but the value of products of these establishments formed only 32.5 per cent. of the total value of products reported for the combined industry. The eight establishments engaged primarily in the manufacture of glucose formed only 6.8 per cent of the total number of establishments, but the value of their products represented 67.5 per cent. of the total value of products for the industry.

The establishments in the industry as a whole in 1909 gave employment to an average of 5,827 persons, of whom 4,773 were wage earners, and paid $4,079,722 in salaries and wages.

The capital invested amounted to $38,866,419. The cost of materials used in the industry as a whole in 1909 was $36,898,771, and the total value of products was $48,799,311.

A striking feature of the industry, as indicated by the figures, lies in the fact that while the number of establishments was less in 1909 than in 1879, the value of products was more than four times as large, thus showing a greatly increased productive power for the average establishment.

As measured by the value of products, Illinois was the most important state in the industry in 1909, followed by Iowa, New Jersey and Indiana, in the order named. In the five-year period, 1904-1909, there was an increase of only two per cent. in the number of wage earners employed in the industry. On the other hand, the value of products increased 49.5 per cent. in five years. The wide discrepancy between the relative advance in value of products and in number of wage earners employed is doubtless indicative of improved processes used in manufacturing the products.

The average number of persons engaged in the glucose and starch industry during 1909 was 5,827, of whom 4,773, or 81.9 per cent., were wage earners. Nearly all the wage earners (98.1 per cent.) reported for the industry as a whole were employed in establishments where the prevailing hours of labor were sixty or more per week.

Form of Ownership

In 1909 of the total number of establishments 46.6 per cent were under corporate ownership, as compared with 40.7 per cent. in 1904. In 1909 the value of products of these establishments represented 96.2 per cent. of the total, and in 1904 96.8 per cent.

Eleven per cent. of the establishments in the industry as a whole in 1909 manufactured products valued at $1,000,000 or over, such establishments returning 89 per cent. of the total value of products, and 18.6 per cent of the establishments reported products valued at $100,000 or over, as compared with 15.7 per cent. in 1904. The average value of products per establishment in the industry as a whole increased from $233,213 in 1904 to $413,553 in 1909. The average number of wage earners per establishment shows an increase from 33.4 in 1904 to 40.4 in 1909. In the glucose branch of the industry the average value of products per establishment in 1909 was $4,116,365, and the average number of wage earners 356. In the starch branch of the industry the average value of products was $144,258, and the average number of wage earners 18.

The total expenses in the glucose and starch industry in 1909 were $43,973,558, distributed as follows: Cost of materials, $36,898,771, or 83.9 per cent.; wages, $2,666,483, or 6.1 per cent.; salaries, $1,413,239, or 3.2 per cent., and miscellaneous expenses, made up of advertising, ordinary repairs of buildings and machinery, insurance, traveling expenses and other sundry expenses, $2,995,065, or 6.8 per cent. The unusually high proportion of the total expenses formed by cost of materials and the low proportion formed by wages is explained by the fact that the various processes in this industry are almost entirely mechanical.

Materials and Products

There was an absolute increase of $16,149,475, or a relative increase of 49.5 per cent., from 1904 to 1909 in the total value of products for the industry as a whole, which was $48,799,316. Corn and potatoes are the most important materials used in the manufacture of glucose and starch. Computed on the basis of 56 pounds to the bushel, the 2,240,508,915 pounds of corn used in 1909 was equivalent to 40,009,088 bushels. Computed on the basis of 60 pounds to the bushel, the 210,608,127 pounds of potatoes used was equivalent to 3,510,135 bushels. Wheat flour is also an important material in the industry. The quantity and cost of arrow root and other roots used as material, are combined with that of wheat, the total for these materials in 1909 amounting to 1,940,000 pounds, valued at $21,435.

Glucose (including glucose syrups) is the most important product when measured by value. The value of this product as reported for 1909 was $17,922,514, or 36.7 per cent. of the total value of products for the combined industry. The increase in the value of glucose from 1904 to 1909 amounted to $5,569,898, or 45.1 per cent. Glucose was manufactured in only four states —Illinois, Indiana, Iowa and New Jersey. Corn oil, a product obtained chiefly in the process of manufacturing glucose, increased in value $1,638,302, or 140.7 per cent. The increase in the value of stock feed amounted to $1,567,489, or 35.3 per cent.

A total of 677,535,647 pounds of starch of all kinds, valued at $17,514,823, was manufactured in the United States in 1909, as compared with 356,695,335 pounds, valued at $10,927,538 produced in 1904. The production of corn starch increased 327,684,552 pounds, or 105.3 per cent., in quantity, and $7,084,466 or 79.8 per cent in value in the five-year period. This large increase was due entirely to increased domestic consumption. Starch made from wheat and roots decreased 4,008,255 pounds or 22.5 per cent in quantity, and $395,724, or 35.2 per cent, in value in the five years.

The quantity of potato starch decreased from 27,709,400 pounds in 1904 to 24,873,415 pounds in 1909, or 10.2 per cent., while the value decreased $101,457, or 11 per cent. Potato starch was manufactured in 1909 in Colorado, Maine, Minnesota and Wisconsin, but the principal point of activity was Aroostook County, Maine, where 61 factories reported the consumption of 171,283,746 pounds, or 81.3 per ceut. of the total of 210,608,127 pounds of potatoes used in the United States in the manufacture of starch, and the manufacture of 20,514,277 pounds, or 82.5 per cent. of the total quantity of potato starch reported for the industry. Starch products are used for food, for laundering and sizing, for finishing calico, for thickening colors, and for many other purposes.

Investigations of Blast Furnace Operations*

In a paper before the May 4, 1913, convention of the Verein Deutscher Eisenhüttenleute, Prof. **W. Mathesius,** of Berlin, developed a method of analyzing blast-furnace operations. The method was then applied to records obtained from twenty-eight different blast furnaces, and charts were drawn that show at a glance the interrelations of the most important factors in blast-furnace practice. Following the theoretical analyses, the results of experiments upon the relative reductibility of briquetted, sintered, and unroasted ores, were given.

In developing the method of analysis, it was assumed that all reactions between the charge and the gases except four could be discarded. These four reactions are as follows:

1—The oxidation of the carbon in the coke, in the hearth by the blast to carbon monoxide.

$$C + O = CO.$$

2—The reduction of iron oxides by the carbon monoxide which takes place in the middle regions of the furnace.

$$Fe_xO_y + yCO = xFe + yCO_2.$$

3—The reduction of carbon dioxide by the carbon in the coke. The former may come either from the charge or be generated by a reaction between carbon monoxide and ferric oxide. This reaction can take place only at temperatures considerably above the well-known Boudouard limit of stability; that is, at least as high as 900° or 1000° C. The equation is:

$$CO_2 + C = 2CO.$$

4—The catalytic dissociation of carbon monoxide which takes place in the uppermost regions of the furnace

$$2CO = C + CO_2.$$

This last reaction can only take place when the substances are considerably above the previously mentioned stability conditions, both as to quantity and temperature; that is, the temperature must be from 400° to 600° C. Above 600° C. the speed of the reaction becomes so slow that it has little practical bearing upon the blast-furnace operation.

Of the four reactions mentioned above, the first, second and fourth are useful, while the third is detrimental to the economic working of the furnace.

Reaction 4 is favorable to economical operation on account of the restoration of carbon that would otherwise be lost and works in direct opposition to the harmful reaction 3. In considering the performance of the furnace as a whole, reactions 3 and 4 can be assumed under favorable conditions to compensate each other and, therefore, in analyzing the performance of a blast furnace, reaction 4 can be left out of consideration entirely, while reaction 3 is only considered under conditions where it is not completely compensated.

It is possible for reaction 4 to become harmful under certain unfavorable conditions; for instance, if it takes place to an abnormal degree, the amount of carbon dust may become so great that it seriously increases the resistance of the charge to the passage of the gases.

Below we give the equations used in the mathematical analyses of data received from the various furnaces:

LIST OF SYMBOLS

$m = \frac{(CO_2)_g}{(CO)_g}$ = the weight ratio of the carbon dioxide to carbon monoxide in the waste gases.

$m' = \frac{(CO_2)_{g'}}{(CO)_{g'}}$ = the volume ratio of carbon dioxide to carbon monoxide in the waste gases.

Further, in kg. per kg. of pig iron,

C = the total carbon from the coke.

C_x = the carbon of the coke consumed by harmful reactions (the dissociation of CO_2 in the shaft and the reduction of Fe_2O_3 in the hearth which in this analysis are considered as identical because they produce the same effects upon the consumption of coke and the heat performance of the furnace).

C_{ez} = the carbon in the CO_2 of the ore and the flux.

C_{fe} = the carbon taken up by the iron.

*Translated in abstract from Stahl und Eisen, Sept. 4 and 11, 1913

C_{mn}, Si, P = the carbon taken up by the reduction of MnO, SiO_2, P_2O_5 in the hearth.

O_e = the oxygen of the oxide of iron in the ore.

O_l = the oxygen brought by the blast in the hearth.

O_b = oxygen from the charge contained in the waste gases.

$(CO_2)_g$ = the carbon dioxide in the waste gases.

$(CO_2)_{ez}$ = the carbon dioxide of the ores and the flux.

$(CO_2)_d$ = the carbon dioxide formed by reduction of the ore with CO, which is again dissociated by C.

$(CO_2)_y$ = the carbon dioxide formed by reduction of the ore with CO.

$(CO_2)_{ezu}$ = that portion of $(CO_2)_{ez}$ which passes up the shaft intact.

$(CO_2)_{ezz}$ = the $(CO_2)_{ez}$ which is decomposed in the shaft.

$(CO)_g$ = the carbon monoxide in the waste gas.

$(CO)_{fe_2O_3}$ = the carbon monoxide produced by the direct reduction of iron ore by C.

The general equations used in the analysis are given in full below:

Carbon Dioxide Equation.—The amount of CO_2 in the waste gases, not including that which comes from the ore mixture and is not reduced by the carbon from the coke in its passage upward, must equal the amount of CO_2 formed from CO by the reduction of ore, less that amount which is again dissociated by the carbon of the coke according to reaction 3.

(1) $$(CO_2)_g - (CO_2)_{ezu} = (CO_2)_y - (CO_2)_d.$$

The carbon dioxide of the ore mixture is split up as follows:

(2) $$(CO_2)_{ez} = (CO_2)_{ezu} + (CO_2)_{ezz}.$$

Carbon Equation.—The carbon from the coke, less that which has gone over to the iron as carbide and plus that which comes from the ore mixture, must be equal to the amount of carbon present in the carbon dioxide and the carbon monoxide of the waste gases.

(3) $$C - C_{fe} + C_{ez} = 3/7\ (CO)_g + 3/11\ (CO_2)_g.$$

Oxygen Equation.—The oxygen of the ore forms CO_2 with CO in the shaft of the furnace, or CO with C in the hearth.

(4) $$O_e = 4/11\ (CO_2)_y + 4/7\ (CO)_{fe_2O_3}.$$

The carbon which enters into the harmful reactions must equal the carbon which is gasified according to $(CO_2)_d + (CO_2)_{ezz}$, together with the carbon which reacts in the hearth forming $(CO)_{fe_2O_3}$.

(5) $$C_x = 3/11\ [(CO_2)_d + (CO_2)_{ezz}] + 3/7\ (CO)_{fe_2O_3}.$$

The expression $(CO_2)_{ezz}$ may be worked out from equations 1 and 2. Subtracting equation 1 from equation 2, we have by proper transposition:

$$(CO_2)_{ezz} = (CO_2)_{ez} - (CO_2)_g - (CO_2)_d + (CO_2)_y.$$

This value when substituted in equation 5 gives:

(5a) $$C_x = \frac{3}{11}[(CO_2)_{ez} - (CO_2)_g - (CO_2)_y] + \frac{3}{7}(CO)_{fe_2O_3}$$

For developing the values $(CO_2)_g$ and $(CO_2)_y$ the equation is written as follows:

(5b) $$C_x = \frac{3}{11}(CO_2)_{ez} - \frac{3}{11}(CO_2)_g + \frac{3}{11}(CO_2)_y + \frac{3}{7}(CO)_{fe_2O_3}.$$

For obtaining $\frac{3}{11}(CO_2)_g$ we have from equation 3:

$$C - C_{fe} + C_{ez} = \frac{3}{7}(CO)_g + \frac{3}{11}(CO_2)_g.$$

By substituting $(CO)_g = \frac{1}{m}(CO_2)_g$ we find:

$$C - C_{fe} + C_{ez} = \frac{3}{77\,m}(CO_2)_g \cdot (11 + 7\,m),$$

hence

(6) $$\frac{3}{11}(CO_2)_g = 7\,m\left(\frac{C - C_{fe} + C_{ez}}{11 + 7\,m}\right)$$

$\frac{1}{11}(CO_2)_y$ is worked out from equation 4 thus:

$$O_e = \frac{4}{11}(CO_2)_y + \frac{4}{7}(CO)_{fe_2O_3}$$

hence

(7) $$\frac{1}{11}(CO_2)_y = \frac{1}{4}O_e - \frac{1}{7}(CO)_{fe_2O_3}.$$

Substituting equations 6 and 7 in equation 5b, we have:

(5c) $$C_x = \frac{3}{11}(CO_2)_{ez} - \frac{7\,m}{11 + 7\,m}(C - C_{fe} + C_{ez}) + \frac{3}{4}O_e.$$

According to the molecular ratio $CO_2 = \frac{11}{3}C$, hence

$$\frac{3}{11}(CO_2)_{ez} = C_{ez}.$$

This, substituted in equation 5c, gives:

(5d) $$C_x = \frac{3}{4}O_e - \frac{7\,m}{11 + 7\,m}(C - C_{fe}) + \frac{11}{11 + 7\,m}C_{ez}.$$

It should be noted that the above equations are purely theoretical, and possess a perfectly general applicability, since no assumptions were made to fit any special local conditions.

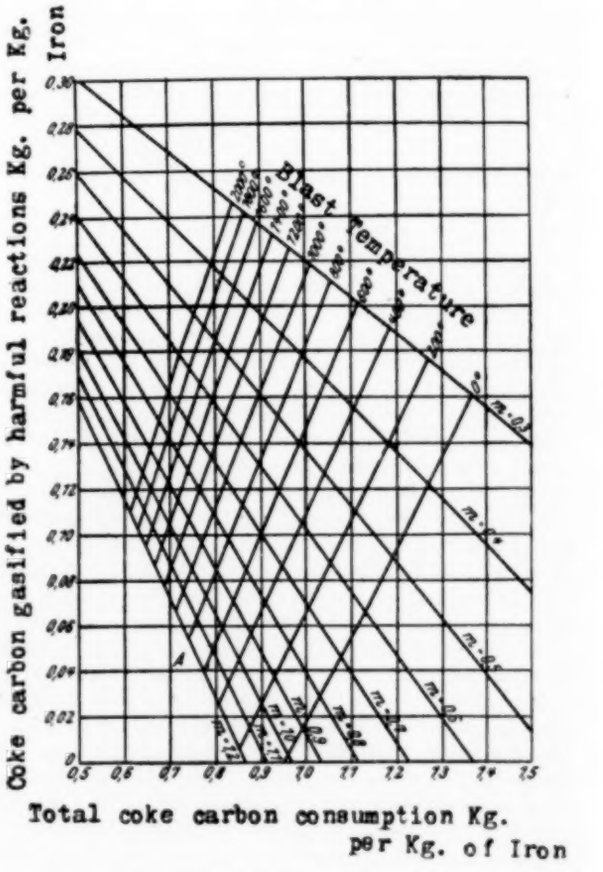

Coke carbon gasified by harmful reactions Kg. per Kg. Iron
Blast temperature
Total coke carbon consumption Kg. per Kg. of Iron
A

FIG. 1—DIAGRAM OF BLAST FURNACE OPERATIONS

If the CO_2 and CO are expressed in volumetric percentages instead of weight, the equation 5d would be written

(5e) $$C_x = \frac{3}{4}O_e - \frac{1}{1 + m'}[m'(C - C_{fe}) - C_{ez}].$$

The equation for C_x make it possible, wherever O_e, C_{fe} and C_{ez} are known, to construct a chart which gives a perfectly clear picture of the relations between C, m and C_x in the operation of any blast furnace. By assuming different values for m Fig. 1 has been thus constructed for operation with basic pig iron which has the following average composition:

C	3.70 %
Mn	1.734 %
P	1.713 %
Si	0.41 %
Fe	92.704 %
	100.00 %

Assuming a blast temperature of 600° C., and calculating the heat balance for the process, the total loss through cooling

water and radiation is found to amount to 24.48 per cent of the total heat supplied.

Now, if it is assumed that the loss remain practically constant for different points in the chart, the blast temperature may be inserted as unknown in the heat balance equation and determined for different values of m and C, the results being represented by lines drawn across the first ones.

The completed diagram or chart in Fig. 1 illustrates the intimate connection existing between coke consumption, blast temperature and the ratio of CO_2 to CO in the waste gases. Furthermore, it shows that with a constant blast temperature the changes in furnace operation must take place along that particular temperature line, thus determining the relations between the other variables and plainly indicates the great effect of the m or "CO_2 to CO ratio" upon the coke consumption, because the other variables remain fairly constant.

A cursory examination of this chart reveals the great importance of keeping continuous records of the m ratio in the operation of a blast furnace. The chart further shows that when working at the maximum m ratio (of CO_2 to CO) the lowest coke consumption is reached, and it may be interesting to trace the reason for this effect.

Naturally the amount of CO_2 in the waste gases is higher, the easier the reduction of the ore; that is, the greater the amount of reduction that takes place in the furnace by action of the gases, or conversely the ratio of CO_2 to CO is smaller, the greater the amount of ore which is directly reduced in the hearth by action of the carbon.

The value of the ratio of CO_2 to CO is also dependent in a very marked manner upon the regularity of working and uniformity of the furnace charge which for best conditions should offer a uniform resistance of the passage of the gases, thus making a uniform distribution and flow.

The point A in Fig. 1 indicates the conditions of practical operation in the particular case which was investigated. This furnace was charged with friable ores, and, therefore, had to be driven slowly in order to avoid trouble, which is indicated by a study of the diagram. On the one hand, the CO_2 to CO ratio is extraordinarily high (1.07), while, on the other hand, the carbon C_x available for compensation of the harmful reaction is relatively small. From these two facts it is evident why, with a yield of only 40 per cent, it is possible to get along with a coke carbon consumption of only 800 kg. per metric ton of iron. The carefully obtained heat balance sheet showed, as mentioned before, a loss of 24.48 per cent in cooling water and radiation.

Realizing that the cooling water and radiation losses for a given temperature vary only slightly for a given kind of iron, whether the operation is conducted fast or slow, it is evident that a furnace of given dimensions and operating conditions which produced double the amount of iron per ton of raw material will have half the cooling water and radiation losses per ton that the same furnace would ordinarily have. Therefore, it follows that blast furnaces should be operated at as high a speed as practicable.

Practical experience in different places, as well as these experiments, have shown that attempts to speed up the furnace operation always lead to the production of white cold pig, and this is true because of the small amount of C_x carbon available for counteracting the disturbances due to non-uniform settling of a friable charge and to the quantities of ore arriving unreduced in the hearth of the furnace.

The operation of a furnace under these conditions is extremely difficult because the varying quantities of unreduced ore which enter the hearth cause large variations in the hearth temperature.

In the case discussed above it appears difficult to determine the best conditions for operation because the furnace must be driven slowly on account of the friable nature of the charge, and this method of operation entails relatively large heat losses, a small coke consumption, and also small amounts of C_x carbon available for harmful reactions in the hearth.

The possibility of choosing more favorable conditions is shown in Fig. 2, which is constructed under the assumption that a large portion, especially of the friable portion of the charge, is briquetted. The effect of this, as shown by experience, is to permit a considerable acceleration of the process without introducing irregularities into the working of the furnace and therefore the heat losses are reduced. In Fig. 2, at point B, it is assumed that they amount to 13 per cent. This sets free considerable heat which is available to counteract the harmful reactions. Due to this, the C_x carbon exceeds 0.14 per kg. of iron, which causes a change in the operating point on the diagram corresponding to a reduction of the CO_2 to CO ratio which goes down to 0.6. This reduction, however, makes it possible to use a hotter blast and thus to bring about a considerable saving of coke, for by raising the blast temperature to 800° C. the operating point may be transferred to C, as shown in Fig. 2.

This method of analysis also shows that increasing the speed

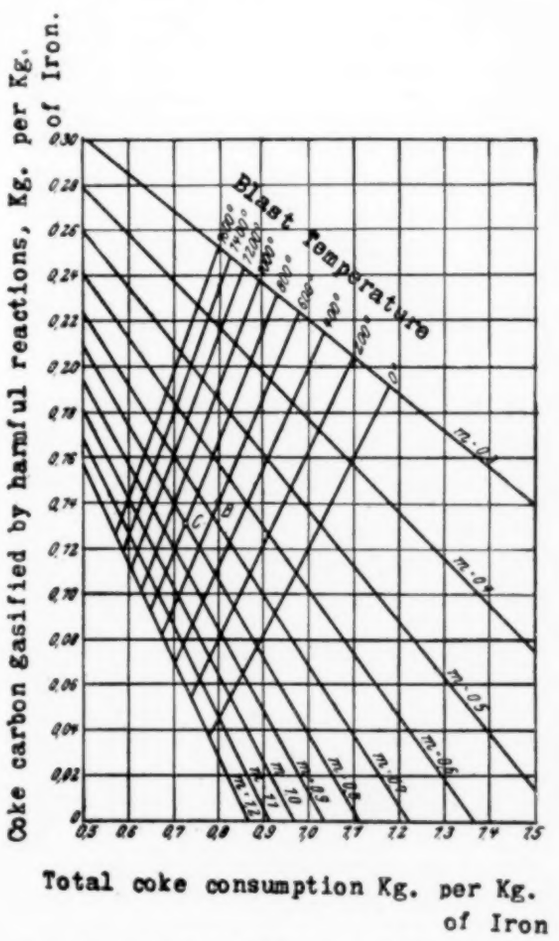


FIG. 2.—DIAGRAM OF BLAST FURNACE OPERATIONS

of operation has the beneficial effect of considerably reducing the amount of blast required per ton of iron. In equation (3) the oxygen required for operation was given. If we substitute in this the values $(CO)_g$ and $(CO_2)_g$, as given in equations (9) and (10), we obtain by simple transposition:

$$O_l = \frac{4\,(11 + 14\,m)}{3(11 + 7\,m)} - (C - C_{fe} + C_{ez}) - O_b.$$

Substituting in this equation the value $m = 0.1$, we have the oxygen required as

$$O_{l_1} = 1.41\ (C - C_{fe} + C_{ez}) - O_b.$$

If we substitute $m = 1$ the oxygen required is

$$O_{l_2} = 1.85185\ (C - C_{fe} + C_{ez}) - O_b.$$

The ratio for the two cases works out:

$$\frac{O_{l_1}}{O_{l_2}} = \frac{1.41}{1.851} = 0.762.$$

In studying the literature of the subject it was found that

widely varying values were assigned to the losses of blast-furnace operation. Indeed, in some cases heat balances were given in which there was more heat on the outgoing side than on the incoming. Consequently, in order to obtain a reliable basis for the assumption for radiation and cooling water losses, it was necessary to calculate a very large number of heat balances for different blast-furnace operating conditions and for this purpose the author obtained suitable data from twenty-eight different furnaces and subjected them to calculations, the results of which are set forth in tabular form. The sum and substance of a careful analysis of these calculations shows that the economy of operation could be greatly increased if the ore could be introduced into the furnace in such a condition as to facilitate its rapid reduction; that is, in briquetted form.

On the other hand, it is very important to note that the total carbon consumption per ton of iron under high-speed operating conditions is not greater than in cases where the operating speed is less and the ratio of CO_2 to CO much more favorable. This is because in furnaces operating at higher speed there is a marked reduction in the radiation losses which offsets a higher coke consumption as compared with the slower furnaces.

In order to obtain a clear idea of the relation between the speed of operation of a furnace and the heat losses a diagram shown in Fig. 3 was developed. In this diagram it was assumed

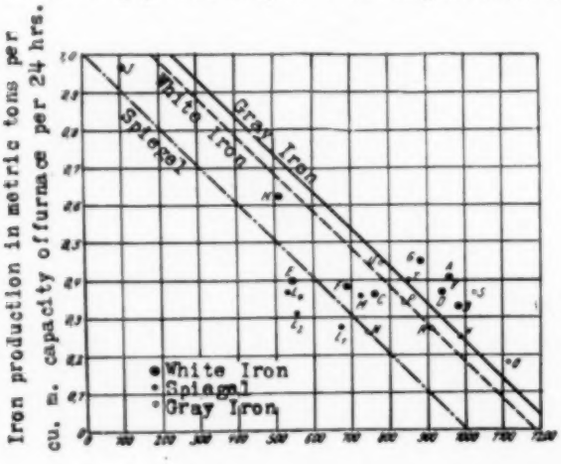

Iron production in metric tons per cu. m. capacity of furnace per 24 hrs.
White Iron
Spiegel
Gray Iron
Heat loss per metric ton of Iron in heat units (Kg. calories)

FIG. 3.—DIAGRAM SHOWING HEAT LOSS BY RADIATION

that the heat losses are inversely proportional to the speed of operation. The points of the diagram were taken from actual calculations. The diagram in Fig. 3 shows that the speeds used in current practice to-day correspond to a loss of approximately 1000 kg. calories per metric ton, and that by speeding up the operation they might be reduced to at least 50 per cent of their present value.

Unfortunately, at present there are only a few cases where briquets have been used for a sufficiently long time under proper conditions of operation to enable the actual saving in fuel to be determined. Consequently, there is not much data from actual practice available.

However, the author was able to present figures obtained in practice by Gustav Gröndal from his experience in Swedish iron works, where by the introduction of briquetted ore he was able in one instance to reduce the charcoal consumption from 1314 kg. per ton to 1000 kg. per ton by using a charge composed of 56 per cent briquets, and in this instance the iron content of the charge was increased from 56 or 57 per cent up to about 62 per cent.

In another case which Mr. Gröndal relates, the charge was made up entirely of briquets, and in this instance the charcoal consumption was only from 560 to 585 kg. per ton of iron.

Such favorable results from the use of briquets in the operation of blast furnaces can only be realized by careful attention to all the conditions that are involved, and operating the furnace in a uniform manner and in such a way that by far the greater portion of the charge is reduced to metal by the gases.

Exhaustive experiments bearing upon the reduction of iron oxides by gases have been conducted at the Technische Hochschule in Charlottenburg, and the detailed results of these investigations will form the subject of a special thesis. However, the nature of the results is plainly shown in the accompanying diagrams.

In these experiments the briquets were tested in quantities of a few hundred grams, each broken into pieces of about nut size. These were heated to different temperatures in an electric furnace and subjected to the action of a stream of illuminating gas for several hours. Under this treatment, depending upon the nature of the briquet, varying amounts of the ore were reduced to metal or ferrous oxide.

The results are shown in Figs. 4 to 18, from which it is seen that briquetting makes it possible to transform practically all of the iron oxides, contained in the ore, into metal. On the other hand, while sintered material must be subjected to the action of the gases for a much longer period, in order to reduce the ferric oxide to metallic iron, the ferrous oxide cannot be reduced by the gases. In some cases the ferric oxide in the sintered ore was only partially reduced to metal, while the rest was transformed to ferrous oxide.

It is noteworthy that the briquetted material gave more favorable results than the lump ore, the cause of which probably lies in the physical structure of the briquet, which, being more porous, is more susceptible to the action of the gases. Figs. 8 and 18 demonstrate this very plainly. Fig. 8 represents a piece of solid magnetite, while in Fig. 18 the same ore was pulverized and briquetted by the Scoria process.

These results show the great importance of using briquetted material in blast furnace operations. By substituting briquets for friable ores, and ores which are difficult to reduce, such as magnetite, it is possible to operate a given furnace at a considerable higher speed, thus increasing its daily capacity and decreasing the percentage losses in cooling water and radiations, the result of which is a considerable saving in coke.

* * *

In discussing Professor Mathesius's paper, Professor **Simmersbach**, of Breslau, made the following remarks:

The results of Professor Mathesius's investigations show that sintered ores are not well adapted to indirect reduction. I fully agree with his conclusions, yet it is nothing new. About ten years ago I brought out these same facts in connection with studies made by me upon sintered iron ore containing manganese. It was shown that with the same manganese content in the sintered ore charge there was less manganese reduced and entering the iron and that for a certain percentage of manganese in the iron additional manganese had to be added to the burden, which naturally caused an increase in coke consumption.

The speaker cannot agree with Professor Mathesius in his assumption that reactions (3) and (4) may be considered as practically offsetting one another. Theoretically this might be permissible, but practically considered, both of these reactions must be taken into account. According to the first reaction, CO_2 is taken from the carbon of the coke, while in the second one carbon is returned; however, not to the coke, but to the ore. The effect of this absorption is to cause the ore to swell, become more porous and thus increase its reductibility by the gases, and this is most important in indirect reduction. According to tests made by Laudig at Buffalo, the carbon absorption by the different ores is as follows:

Mesabi ores	21.61 per cent of the ore weight
Soft hematite ores	13.82 per cent of the ore weight
Hard hematite ores	7.52 per cent of the ore weight
Hard crystalline hematite ores	3.08 per cent of the ore weight
Iron slags	0.38 per cent of the ore weight
Magnetites	0.00 per cent of the ore weight

In absorbing this amount of carbon the Mesabi ores swelled to double their normal size and became very much more porous. I have here a piece of magnetite taken out of the hearth

of a blast furnace, which 2 mm. below its surface is still in its original unaltered condition. This incomplete reduction is due to the density of the magnetite.

In the mathematical analysis of blast furnace operations as to the deposition of carbon, it must be borne in mind that the burning of coke according to reaction (4) produces 50 per cent more heat than direct burning; this is demonstrated in the following calculations based on furnace gas containing 1 CO_2 to 2.25 CO.

(a) *Direct Burning.* ($C + O = CO$ and $CO + O = CO_2$).

1C	develops by burning to CO_2....	8080 heat units
2.25C	develops by burning to CO	5557.5 heat units
3.25C	develops by burning.....	13637.5 heat units
1C	develops by burning.....	4196.5 heat units

(b) *According to Reaction* (4), ($C + O = CO$ and $2CO = CO_2 + C$).

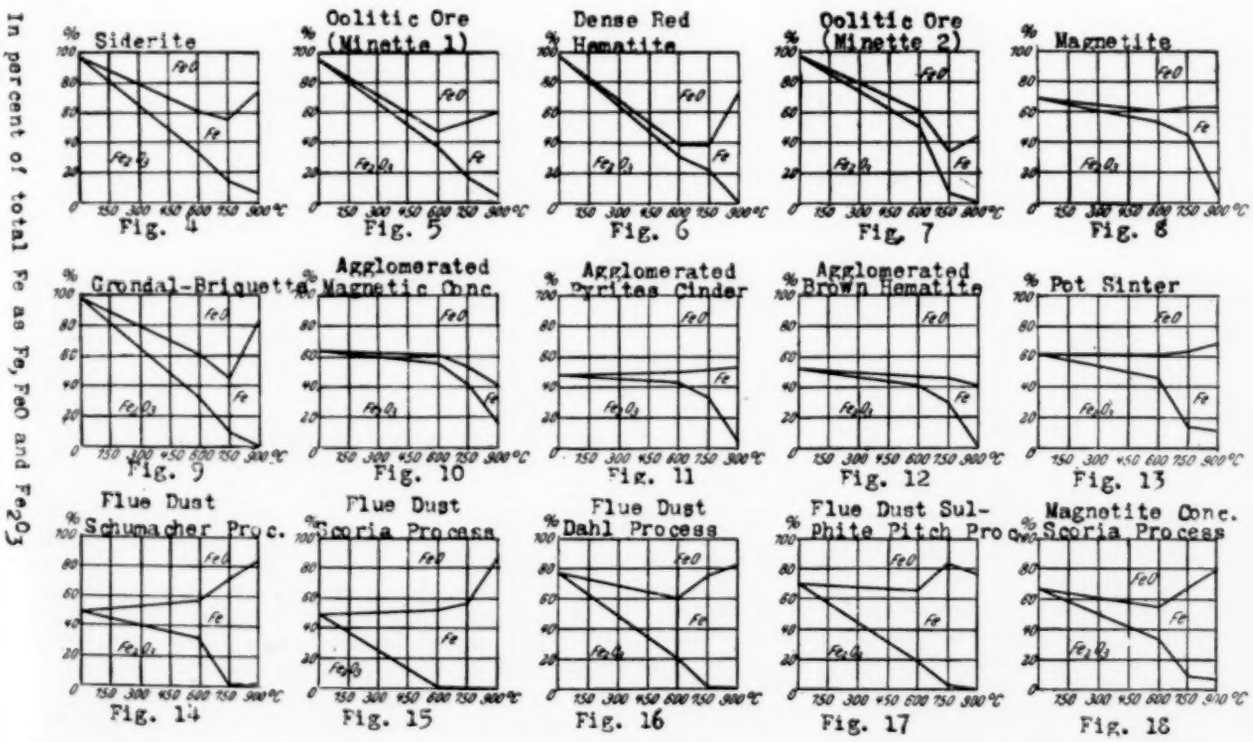
2C	develops by burning to 2 CO .... —	4940 heat units
$2CO = CO_2 + C$		
1CO	develops by burning to CO_2....	5610 heat units
1CO	absorbs by the reduction to C..	2470 heat units
		3140 heat units
1C	gives according to direct burning.	4196 heat units
2C	develops	12276 heat units
1C	develops	6138 heat units

Pb	1.21
Cu	0.51
Al_2O_3	0.84
CaO	0.96
MgO	0.94
Sulphur as sulphide	2.27
P	0.114

The balance mostly oxygen.

In the briquetting of flue dust a number of important factors are to be considered, namely, the variable chemical composition, the quantities of raw, sintered and melted particles composing the dust and also the coke fines therein contained. The age of the dust and the place from which it was obtained is also important, as fresh dust always contains hydraulic binders. If left for any length of time in the open or in flues distant from the furnace, subjected to the influence of moisture or carbon dioxide, the binding constituents will become hydrates and carbonates. Dr. Schumacher first discovered the practical significance of the last named phenomenon, and thereby developed his briquetting process.

When briquets are tried out on the furnace, it is customary to replace a certain amount of ore by as large a quantity of briquets as possible. If there is an improvement in the operation of the furnace, the briquets are credited with the same. If, on the contrary, the operation of the furnace is impaired,



FIGS. 4 TO 18.—REDUCTION OF ORES SINTERED AND BRIQUETTED MATERIAL. FIGS. 4 TO 8.—ORES. FIGS. 9 TO 13.—SINTERED ORES. FIGS 14 TO 18.—BRIQUETTED ORES AND FLUE DUST

Because of the lack of time, I refrain from further criticism as to the author's method of calculation, since his conclusions have for years been found correct in actual practice.

Referring to the results of experiments as plotted in Figs. 4 to 18, it should be noted that the various processes cannot be compared with each other unless the same raw material is used in all cases. Considering particularly a flue dust briquet and its reductibility the following analysis was obtained from a partly reduced Schumacher briquet containing 25 per cent metallic iron:

Total carbon	7.46
of which coke, graphite.....	4.88
SiO_2	6.55
Fe	61.06
Mn	0.40
Zn	6.70

it is a question whether this is directly chargeable to briquets, or only indirectly. When using briquets on a furnace the blast is of primary importance, and the pressure, the amount and their temperature must carefully be considered. Although sintered material is more difficult to reduce than briquets, and therefore less advantageous, yet there may be unusual conditions under which sintered ores, with correspondingly high temperatures and certain ore mixtures, will give better results than briquetted material.

In the author's experiments better results were obtained from briquets than from lump ores. A reason for this, besides the difference in density, is due to the coke amounting to from 10 to 25 per cent, and which is distributed evenly in the briquets. It is my opinion that the coke in the briquet should be figured as coke in the calculated charge, as to the coke consumption.

Professor **Osann** made the following remarks in the discussion:

The author's temperatures used in test are too low to be of greatest practical value, as they do not indicate what takes place in the hottest part of a blast furnace.

Professor Osann takes issue with the author on the point that the most easily reducible ore mixtures are always the most desirable ones. Practice does not, in all cases, bear this out. Ores that are difficult to reduce are often added to the charge in order to improve the performance of a furnace by counteracting certain difficulties that occur when easily reducible ore mixtures are used.

It is well known that ores that are difficult to reduce are very desirable when first-class iron for casting is wanted. The very easily reducible Lake Superior ores are reduced in American furnaces with no less an amount of fuel than are the ore mixtures of the lower Rhine, which are rich in magnetite. The reason of this is that easily reducible ores pass so quickly through the furnace that the saving in fuel is offset.

The blast furnaces in America use at least as much fuel as the furnaces in Germany under similar conditions, and probably the American furnacemen would be well pleased if their ores did not reduce so easily.

It is also well known that in Siegerland and in Steiermark red spar is not used alone but about one-third of raw spar is added. By this means the disturbances caused by the easily reducible ore are practically eliminated.

The cause of these phenomena we do not exactly know at the present time. We do know, however, that in blast furnace operation all the different factors work together and that no one of them can be considered by itself.

As far as deciding which briquetting process should be used, no other way is open except experimentation, and these experiments must be continued over considerable length of time. Professor Osann advises that, without any prejudice, experiment should be conducted with both sintering and briquetting processes and decision should be made entirely in conformity with results obtained.

Some processes should be dropped out at once as being of too little advantage or of too great a cost.

Sinter is often compared with slag because of its resistance to reduction; however, this is not entirely permissible, because slag consists principally of silicates while sintered material is produced by gangue going into solution in iron oxides without the necessary melting.

When flue dust briquets, made by the use of a binder, sink into the furnace, they soon behave in some similar way to sintered material.

Professor Osann recommends a program for exhaustive experiments to prove the value of briquetting and then closes by criticizing the use of temperatures below 900° C. in the tests and the assumption of a constant heat loss in the analysis. He hopes, at some future time, to be able to discuss in detail the effect of speed of operation upon heat losses.

Condenser Tube Corrosion

Past and Future Work of the Corrosion Committee of the British Institute of Metals

By Sir Gerard A. Muntz, Bart., and Professor H. C. H. Carpenter, M.A., Ph.D.

The (British) Institute of Metals Corrosion Committee, which consists of the following members: Sir Gerard A. Muntz, Bart., chairman; Professor H. C. H. Carpenter, M.A., Ph.D., Hon. Secretary; Dr. G. D. Bengough, M.A., Hon. Investigator; Mr. L. Archbutt; Engineer-Rear-Admiral G. G. Goodwin, R.N.; Professor A. K. Huntington, Assoc. R. S. M.; Mr. J. T. Milton; Mr. A. Philip, B.Sc., Assoc. R.S.M.; Sir W. E. Smith, C.B.; Mr. L. Sumner, M.Sc., and Professor T. Turner, M.Sc., has now been at work for some three and one-half years.

Under its guidance two reports have been published by Dr. G. D. Bengough, who has carried out his investigations in the Metallurgical Department of the University of Liverpool.

The first report, issued in January, 1911, and published in Vol. V of the *Journal of the Institute of Metals*, was intended to give a general review of the knowledge as to the corrosion of non-ferrous metals both in its practical and scientific aspects, and is the most comprehensive survey of the subject that has yet been published. On the basis of this report the committee decided to initiate an experimental investigation of the cause or causes of the corrosion of condenser tubes in marine engines.

An appeal for funds was issued and the response was sufficient to enable a two years' research to be undertaken, which formed the subject matter of the Second Report to the Committee presented at the Ghent meeting of the Institute of Metals held at the end of August in the present year.

CORROSION RESISTING TUBES

The experimental work has been carried out both in a special condenser and by means of laboratory experiments. The condenser plant was erected with the object of reproducing as accurately as possible the conditions to which condenser tubes are submitted in actual practice, and the laboratory experiments were designed to interpret the observations made with the larger apparatus, a procedure which has been signally successful in its results. In both cases it was found possible to observe and study "dezincification" which is the usual cause of the failure of tubes in practice. Tubes of four different compositions have been examined under strictly comparable conditions. The ordinary 70:30 tube of commerce has been shown to be far surpassed in resistance to dezincification by tubes containing 70 per cent copper, 28 per cent zinc, and 2 per cent lead, and by those of "Admiralty" composition, *i.e.*, containing 70 per cent copper, 29 per cent zinc and 1 per cent tin.

Muntz metal, 61 per cent copper, 39 per cent zinc, has been shown to be unsuitable for use as condenser tube material.

GREAT IMPORTANCE OF TEMPERATURE

The influence of a variety of "deposits" and "particles" which have hitherto been supposed to cause localized dezincification has been shown to be of only secondary importance, and dezincification has been found to start and proceed rapidly in their absence. *In all cases the rapidity of the failure of the tubes was found to depend principally upon the temperature to which they were subjected.* If corrosion is to be prevented, or at any rate minimized as much as possible, *the temperature in the condenser must be kept low.*

Preliminary experiments have been made on the mode of action of electrochemical protection. This has been found to be intimately connected with the deposition of a continuous layer of calcium carbonate upon the inner surface of the tubes.

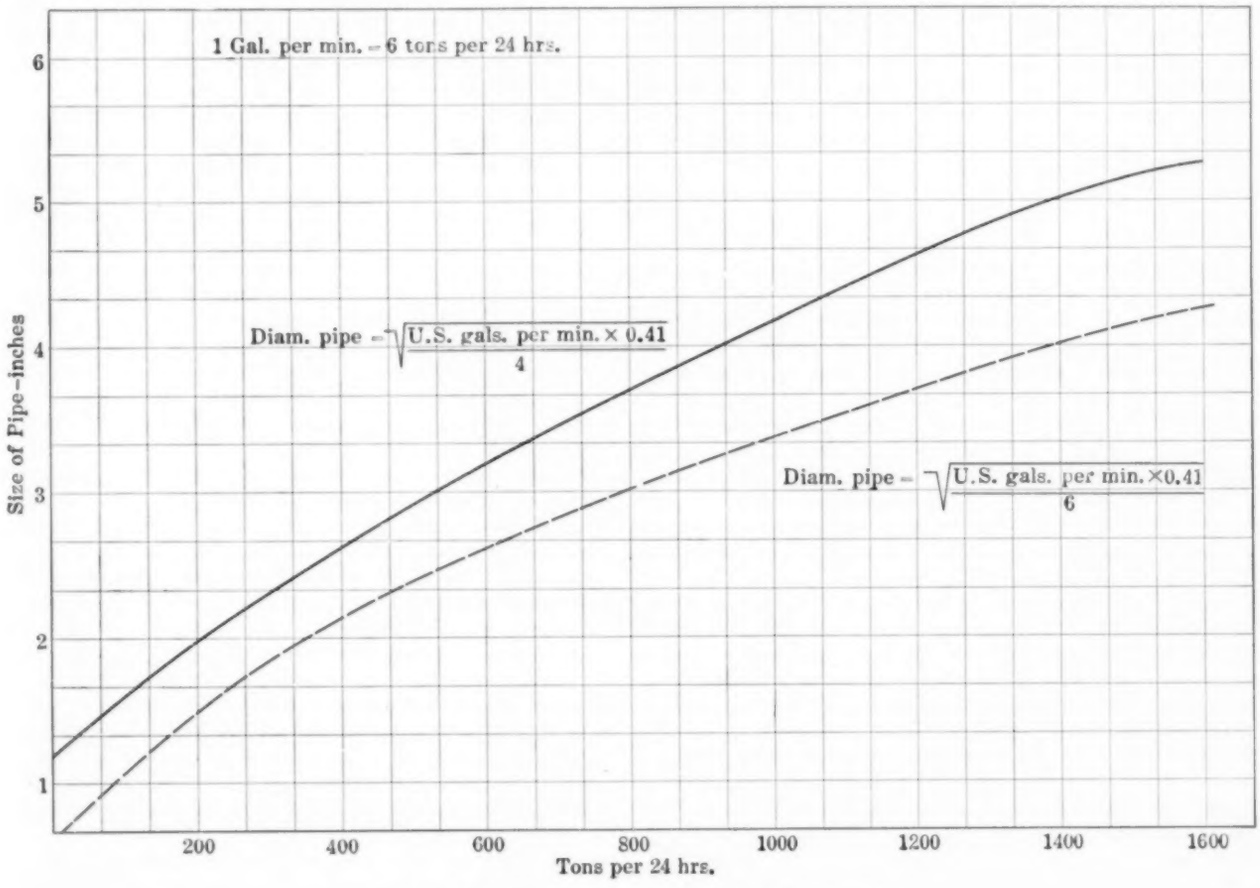
SCHEME OF NEW YORK

The committee have met recently and having obtained sufficient funds to continue their investigations for another two years they have planned a further scheme of work which is now well in hand. In deciding upon the lines of future investigation they have been careful to take into consideration a number of valuable suggestions and criticisms which have been made on the Second Report both in the discussion at Ghent and in the comments of the technical press. The scheme includes a complete time-temperature survey of the conditions under which dezincification can take place and a more detailed study of the temperatures existing in their own and other condensers. Special attention will also be directed to the conditions under which electrochemical protection can be effectively maintained, and in this connection a comparative study of all the principal methods of holding tubes in tube-plates will be carried out. A systematic series of experiments will also be initiated with a set of tubes of a copper-aluminium alloy from which specially interesting results are anticipated.

The foregoing scheme by no means exhausts the lines of investigation which in the opinion of the committee might profitably be undertaken, and if further financial support is forthcoming they will gladly extend their researches in other equally vital directions.

Sizes of Pipe for Cyanide Mills and Industrial Plants

The accompanying curves will be found useful in determining the size of pipes to be used for water and solution in cyanide mills, or in industrial plants where it is necessary to convey definite volumes of liquid through pipes from one point of a process to another. The present chart was drawn by Mr. Justin H. Haynes, and used by him in the design and construction of the Junta mill, near Telluride, Col. The solid line refers to a velocity of 4 ft. per second, and should be used whenever possible. The dotted line is based on a velocity of 6 ft. per second and is not normally used.



SIZES OF PIPES FOR CYANIDE MILLS AND INDUSTRIAL PLANTS

An acid-proof lining for tanks can be prepared from the following formula: 10 per cent litharge, 20 per cent short-fibre asbestos and 70 per cent sand, mixed to a plaster with a 40° Be. solution of silicate of soda. The plaster can be applied as thick as desired. An application of acid after drying will "set" the cement and render it acid-proof.

Refractories in Montana.—Among the raw materials used by the Anaconda Copper Mining Co. for making refractory brick are (1) a plastic clay from Armington; (2) a flint clay from Lost creek; (3) silica, and (4) "Dillon" rock. The silica (3) used for silica brick contains about 97 to 98 per cent SiO_2; lime is used as a binder. The "Dillon" rock runs 99 per cent SiO_2, and is used for sanding molds; it is a fine, dense, white sandstone, not fit for making silica brick. Analyses of the two clays are given below.

	Plastic	Flint
SiO_2	55.7%	76.9%
FeO	1.1	0.9
MgO	0.5	0.0
Al_2O_3	30.8	17.3
CaO	0.2	0.0
Alkali	2.1	——
H_2O, etc.	10.2	——

Ozone

By A. Vosmaer, Ph.D.

In ordinary oxygen two atoms are combined to one molecule, but it takes three atoms to form a molecule of ozone. Different names have been given to this particular gas. Some people have called it activated oxygen, others have given it the name of condensed oxygen, some have spoken of electrified oxygen and the name polymerized oxygen has also been suggested.

None of these suits the purpose in a strict way. It is better to distinguish sharply between substances of different properties. In fact, Ostwald who defines a body as a state of things differing from its surroundings, considers allotropic modifications as different substances and this seems very sensible. It is really too bad to have different sets of properties for one and the same substance in different modifications, such as red and yellow phosphor, or diamond, graphite, and carbon, or sulphur in all its varieties. Is not it much more plausible to consider them as really different substances?

The fact that chemically they closely resemble each other does not entitle us to consider them as one and the same, for it is not only the chemical properties which determine a substance, and even in that respect we are not always so very sure about it.

The names "activated" and "condensed" oxygen are wrong altogether. Ozone is more active than oxygen because one, or all three of its atoms, are holding each other much less firmly than the two in oxygen. The well-known oxygen in *statu nascendi* is none the less as active as ozone. One should not lose sight of the fact that if we speak of oxygen we have in mind the gas in its molecular, (two-atomic) state and not in its atomic state. Atoms are altogether different in their behavior from molecules, so we must guard ourselves against mixing them up. This consideration also condemns the word "condensed," because there has been no condensation of O_2 but of O.

The prefixes "electrified" and "polymerized" do not please me

either, the first because it is too vague, the second for the same reason as above, that ozone is no polymerization of oxygen, is not $(O_2)^n$ but is O_3.

What other name shall we give it then? Well, if we call it ozone there is no need for another name. Its composition has been fixed to be O_3 its molecular weight being 48.[1]

A great difficulty for philosophical consideration is that we know all substances by their molecular state while in chemistry we deal with them in their atomic state. This cannot be altered for the present. It has to be endured as long as it cannot be cured.

MANUFACTURE OF OZONE

Ozone is difficult to make. It is an endothermic compound, which means that it absorbs heat when forming. How much has not been settled yet. The amount seems to be in the neighborhood of 30,000 gram calories per gram molecule. But unfortunately the temperature of formation and that of decomposition lie very close together, so that, although heat is wanted, it should be kept away as well. In other words as soon as the ozone is formed it should be taken out of the heat zone.

Practically this means that ozonators work best at the lowest possible temperature. There is always quite enough heat in the discharge itself for the formation of the ozone and all that is formed must be quickly drawn away, out of the influence of the discharge which also has the tendency to desozonise.

These circumstances present interesting cases of equilibrium reactions. When working with air it is hardly possible to get a higher concentration than about 30 grams per cubic meter (containing about 260 grams of oxygen) and this figure does not run up so very much higher when using pure oxygen instead of air.[2]

Ozone may be made chemically, mechanically, and electrically and in different ways, but most of them have not the slightest importance from a commercial viewpoint.

By way of example we call attention to the chemical method of making oxygen by means of sulfuric acid and permanganate of potassium; oxygen thus made contains some ozone. By way of curiosity we mention the fact that violent mechanical disturbance of the air may cause some very little ozone to be formed. It has been reported that when grinding wheels were being tested for bursting speed and when they were flying to pieces the odor of ozone was noticed.

Among the electrical methods there are two; one in the foreground, one in the background. The latter one is the electrolytic when electrolyzing strongly acidulated water, or rather diluted sulfuric acid (sp. gr. 1, 1), the oxygen liberated is strongly ozonized; up to 17% (according to others up to 25%) of the oxygen may be ozone, the process being carried out at low temperature and with a high current density. If we take something like 70 or 80 grams of oxygen per kw-hour, as a practical figure, the yield in ozone comes up to 20 grams under the very best conditions. The yield is low but not so low as to cut out competition, because the other products, oxygen and hydrogen, may have their share in the business and may considerably reduce the cost.

For the present moment, however, the other method, that of electric discharges though air or oxygen is ahead of all other methods on account of the yield being several times that of other methods. *Commercial* ozonators yield up to 70 or more grams of ozone per kw-hour.

The subject of ozonators is one of importance but is too large to be treated in a few words.

For our purpose it will be best to consider only the very best make. Other systems may have scientific interest, but have no chance of permanent application.

None of the systems in which no solid dielectric is used, has been able to make its way, while the system based upon the use of a solid dielectric has succeeded in holding its original position.

The principle underlying is very simple. Any two conductors with sufficiently high potential difference (say above 6,000 volts) and separated by some solid dielectric will discharge the electricity in the form of the so-called dark, silent or brush discharge. This is the kind that possesses the remarkable property of converting oxygen into ozone. Although this principle is simple enough in itself, the details of the method by which it is carried out into actual being, are very important. Numberless patents have been taken out for different arrangements, but, most of them are stone dead and only few are of value.

Those patents are not basic. They all refer to some arrangement, form of electrodes, shape of dielectric and so on. We shall not discuss them here because this article is intended to show the importance of ozone rather than that of some particular make of ozonator.

As said any arrangement of electrodes separated by a dielectric will do. Not all will do equally well. Some apparatus are excellent in the laboratory, others are good in technical work. Some people want ozone of very high concentration, others again want it at least cost. In some cases reliability is of first importance, for instance, when using ozone for the sterilization of drinking water. In other cases high output per kw-hour is the principal requirement. As we intend to devote a future article to ozonators, we shall not enter into details now.

If we compare the theoretical equivalent of ozone in kilowatt-hours with the actual yield, a very great discrepancy is found. We ought to obtain over 1000 grams and at the most we get 100 grams per kw-hour, that is 10 per cent at the very best, and yet there is little hope for further improvement unless some genius strikes a happy thought. For the present moment one has to be content with such law figures as 10 per cent and in most cases not more than 5 per cent efficiency.

PROPERTIES OF OZONE

The pre-eminent property of ozone is its extraordinary oxidizing power, which from a chemical standpoint, can be best judged from the fact that it oxidizes silver.

Popular pamphlets enumerate a great many substances that will be oxidized by ozone, including such substances as ferro compounds, mercury compounds, etc. Now this is not worth mentioning, since if we once know the tendency of ozone to oxidize, it goes without saying that certain reactions will occur. We need not give such a list which anyway could be enlarged to any length.

What is particularly interesting about this property of ozone is that when it has fulfilled its oxidizing function there remains nothing but oxygen, which, being a gas, can easily be got rid of if so desired. This point should be borne in mind when we have to carry out some oxidizing process because in many cases we want to oxidize but do not want any other reactions such as are likely to occur when we use one of the more common oxidizers.

Peroxide of hydrogen has a similar advantage in so far as what is left in its case is water. It is, however, not so easy to get it in a pure state. It does not keep very long unless in acid solution, but it answers the purpose if pure.

We need not insist upon the fact that very few chemical reactions take place when we just drive oxygen gas into some solution. With ozone this is quite different. Here we have oxygen in *statu nascendi* exempt from other chemicals. It will, therefore, prove a very valuable agent in many chemical reactions.

It is disputable whether or not its remarkable bactericidal power is the same as the foregoing, or if it is another property.

If we stop to realize what it means to sterilize water containing thousands and thousands of bacteria per cubic centimeter by using a quantity of one in a million (that is one gram of ozone to sterilize 1,000,000 grams of water or 1 cubic meter or about 260 gallons), we feel inclined to attribute this abnormal power to some definite and specific action on bacteria. As matters are presently I would not dare, however, to claim such a specific property for ozone. It may be just nothing but ordinary oxidation.

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[1] Leaving out of consideration, for the present moment the facts that partly it may be of more complicated nature, i. e., partly O_4, partly O_6.

[2] Professor Warburg obtained values of over 200 but on very small laboratory scale.

Ozone is said to cause phosphorescence in water. It may be so, I have never seen it. But I happened to watch another very curious phenomenon which went on for a whole day. But afterwards I have never been able to get it again. This unfortunate circumstance has prevented an explanation or even a suggestion.

On one of the hundreds of days that I have been working with ozone and water, once it happened that the sterilizing apparatus, which was made of glass so as to permit watching the operation, turned a violet color.

The sterilizing apparatus was a glass cylinder, 1 ft. in diameter, and 30 ft. high (built up in sections). During all that particular day the regular appearance of that column was changed to a violet color, about the shade of a faint amethyst or the reflection of arc light on a wet street. Probably the phenomenon has no bearing upon ozone but is merely a physical effect due to an intimate and very finely divided mixture of gas and water. The same color may sometimes be observed at the edge of the water line when going at a good speed in a calm sea.

That much for the chemical properties of ozone. The physical properties are of importance when it comes to the making of ozone. Ozone is a colorless gas with a sharp penetrating odor which has often been compared to that of chlorine or phosphorus. However, chlorine smells absolutely different from ozone, and comparing it with phosphorus means comparing it with itself, since the peculiar smell of phosphorus is probably due to slow combustion which is said to generate ozone. Here again I would rather say that ozone smells like ozone.

The gas is very unstable. It takes very much trouble to have that third atom join the two others and it is a very loose connection. It decomposes slowly at ordinary temperatures, but rapidly at high temperatures. Above 270 deg. C. it cannot exist under ordinary circumstances.

At a pressure of 125 atmospheres and a temperature of minus 103 deg. C. ozone becomes liquefied, it is then a dark blue mobile liquid of notable magnetic susceptibility.

Ozone is an endothermic compound, that is it requires heat for its formation, which, however, does not mean at all that the hotter an ozonator the better. On the contrary, the cooler the better.

The property of being very unstable prevents us from being able to store ozone. We hardly can expect ever to be able to sell ozone as a liquid in steel bottles like carbonic acid or so. But there is no need for this, since the market offers many kinds and sizes of ozonators and with them any amount and any concentration can be obtained that may be wanted in the industries.

USES OF OZONE IN DAILY LIFE

The utility of ozone culminates in the purification and sterilization of drinking water. This is a subject of such importance that it is rather difficult to do justice to it while confining the discussion to a few lines, as is necessary in this short article.

It can be done when we are allowed to leave out the proofs of what we are saying. Be it sufficient to say that all statements are based on personal experience, except the experiments with pathogenic bacteria, which have been carried out by the German Imperial Board of Health.

To be very brief, we may just summarize what is known by saying: Ozone when of proper concentration and when judiciously applied will kill any amount of bacteria in water. We tested up to as many as 600,000 per cubic centimeter. Ozone acts selectively in so far as the pathogenic, which are the least resistant, are killed first. The German Board of Health tried mixtures up to 30,000 cholera and as many typhus bacteria, added to ordinary river water and none of those were left after treatment.

Some people do not realize the size and nature of bacteria and feel uneasy about their carcasses, but a little figuring will reassure them.

Suppose some water to contain 100,000 bacteria per cubic centimeter, which is an abnormally high figure. Then there are 375,000,000 in a gallon, their size is about 0.00004 in. their cubic size 0.000000000000064 cubic inch so that the tremendous number, 375,000,000 bacteria per gallon occupy only 0.000024 cubic inch for the whole lot. Now consider a bacterium to consist of about 90 per cent of water, and 10 per cent solid, which solid may contain something like 0.1 per cent of indestructible mineral matter than said 0.000024 cubic inch of bacteria yield 0.000000024 cubic inch of "ashes." Thus it is evident that there is so little matter in bacteria that it matters little whether or not we take it into consideration. Moreover, one has to remember that water usually contains some mineral matter so that after all the last figure should have been expressed in percentage increase and that shows the absurdity of thinking of remnants or cadavers of bacteria, which moreover are not of animal but of vegetable substance.

There is overwhelming evidence as to the absolute bactericidal power of ozone in water, but one *conditio sine qua non* has to be considered, viz., that of competent planning and expert management. Both conditions have very often been overlooked and failure could not but result.

Besides its bactericidal power, ozone has also that of taking away all discoloration, even that caused by organic ferric compounds (the precipitated iron has to be removed afterwards by suitable means). It also takes out any disagreeable smell and taste, giving a taste like fresh spring water, which is probably due to the excess of oxygen.

Everybody who really has had thorough experience with the ozonization of water will have to acknowledge the absolute superiority of ozone as a purifier to any other method. Ozone is the ideal. Nothing more is left to be desired, not even the cost. We may mention here that as 1 gram of ozone can treat 1 cubic meter, or say 250 gallons of water of medium quality, and as one gram ozone of sufficient concentration per cubic meter of air to do the work can be produced by 20 watt-hours, there are required 80 kw. for 1,000,000 gallons of water per hour. The cost of this amount of energy *delivered 24 hours* a day is a figure which varies from place to place, but is by no means prohibitive, as is well known to readers of this journal.

We wish particularly to draw attention to this extremely low figure of cost because many people repeat the verdict "too expensive" without even knowing the cost and certainly without realizing the advantages.

Another reason why we mention the price is that, after all, ozone is expensive in itself. It is because so little of it is needed for *this* purpose that the cost comes so low. For many other chemical processes it is out of the question because of its price. Ozone should never be used where other means are just as good. Its use should be restricted to those cases where other means fail to do the work or fail to do it as satisfactorily.

The next best thing after pure drinking water is pure air and this is another achievement of ozone. While it does not sterilize it when dry, yet it takes away nasty odors, like that of tobacco smoke and partly masks, partly oxidizes products of respiration, perspiration, and other volatile matter. Ozone-treated air cannot compete with real pure air from the mountains or at sea, but it invigorates when being breathed in adequate dose, by all means not too strong. A proportion of about one in a million is quite enough for ordinary purposes.

It does make some difference to the men in an over-crowded office whether or not the air breathed by every one of them several times over is ozonized or not. Anyone who actually has experienced the great difference in sensation will be able to appreciate fully the beneficial effect of ozone. It has been a mistake, however, to attribute to ozone what it really does not do, viz., sterilize the air.

The beneficial effect is unquestionable. Moreover, the experiment is very easy to try. There are many air ozonizers on the market. The energy which these small apparatus require is hardly worth speaking of, as it is of the order of magnitude of that of an incandescent lamp.

USES OF OZONE IN THERAPEUTICS

We must be very short on this subject however important it

may be, as there is very little reliable information to be had on this matter.

There is some evidence that ozone is very beneficial to phthisis patients when in the first stadium. It should certainly be not stronger than in the proportion of one in a million or one in ten million parts of air, when intended for continuous inhalation. This point seems of enough importance to be taken up seriously by some official medical society.

When we consider obesitas as the result of inadequate power of oxidation, the body building up no further than to fat instead of going to a higher step of oxidation, for instance, fibre, tissue, flesh, and so on, ozone might be of help.

The therapy of "eating" oxygen in the form of peroxide of barium and other peroxides seems very poor indeed; it is very probable that through inhaling ozone, of course, in very weak concentration, additional oxygen may be brought into the system.

It is dangerous for a layman to discuss matters of a medical nature. It will be better to make one statement which is easily controllable and this is that the inhalation of ozone causes a rise in the percentage of oxyhæmoglobin when this happens to be below the normal, and this rise accounts for the supposed beneficial effects.

Cases of chlorose, and all that is connected with the blood in a direct way, may be treated successfully with ozone. But my own personal experience does not go further than the increase mentioned in the percentage of oxyhæmoglobin, which suggests or even indicates possible favorable results from proper ozone treatment.

INDUSTRIAL USES OF OZONE

Let us bear in mind what we said about the price of ozone that there never is a chance for ozone where cheapness is the very first question to be considered. But cheapness is not always the main factor. Very often quality comes in as a more important item.

As a bleaching agent ozone has to compete with chlorine. For bleaching paper pulp, however, there are other factors to be considered, like the size of the fibre, which is much better if ozone is used. In fact, when comparing ozone-bleached pulp with chlorine-bleached pulp the difference is striking, the individual fibre being about three or four times as large in the former case. This is worth while considering, for the longer the fibre the more "stuff" can be added to make paper out of it.

The bleaching of wax has been carried out with fair results, but not so striking, as to think much of it. The bleaching of sugar molasses succeeds easily, but at the expense of power of crystallization. This fact does not surprise us, when thinking of the hyper-sensitiveness of sugar to turn to the non-crystallizing kind.

The treatment of milk is a great problem of excessive importance, but the trial of ozone for this purpose leads only to disappointment, since the very tender organic compounds contained in milk, are destroyed long before any of the bacteria are destroyed and this is not surprising.

It may be mentioned here that it was the desire to sterilize milk which in Holland led to the pioneer work of Tindal (Schneller patents). When he saw that ozone would do nothing but harm to milk Tindal was the first to recognize the possibility that ozone might be beneficial to water. I dare say that to Tindal is due the present status of water sterilization by ozone. All that has been done—and that is a great deal—has been the result of his showing the way and full credit should be given to him although usually it is not customary to acknowledge the merit of those who did not succeed themselves but gave the impetus to the success of others. Tindal died some years ago.

The treatment of flax can be carried out with great success with ozone. The process known as rotting consists of having the glue of the stalk destroyed through the development of certain bacteria. This result is desired because when combing the raw flax, particles of glue prevent it from being done so effectively or may cause loss of material.

Ozone-treated flax does not contain any glue or other sticky matter and thus said treatment improves the product to a great extent, and more especially so the yield of fibre.

It was not until very much later that I found the reason of this favorable result.

I found it by chance when trying the effect of ozone on glue, the purpose being a bleaching of it.

It turned out that ozone bleaches glue effectually but, after having been thus treated, glue did not possess any more of sticking property. For gluing purposes it is no longer suitable, but the same phenomenon explains the beneficial effect of ozone in treating flax.

Another vast field of possibilities is found in the industry of oils and fats. When making varnish out of linseed oil, discoloration is hardly to be avoided on account of the higher temperature used. If, however, the oxidation is performed by ozone it can be done entirely at ordinary temperature. The result is a perfectly transparent varnish of high quality—and of higher price, as may be added.

For the manufacture of surrogates for butter use is being made of cottonseed oil. This oil, however, possesses a rather strong and nasty taste; to take that taste away has been tried and it can be done.

Another problem has been the bleaching of santal oil, a very expensive oil, the bleaching of which is done by aid of sunlight or peroxide of hydrogen.

I have tried ozone for this oil and obtained a very good result.

However, these questions are rather more commercial than technical, for the problem turns out to be this: Does it pay to spend extra money on the purification so as to get a higher price for the product? Such questions cannot be treated in a short condensed summary like this.

There are numberless other possibilities for the use of ozone. Very many have been suggested and perhaps some have been tried. But as I did not want to make any statement in this article unless backed by my own personal experience I cannot give information on those other possibilities, not knowing them from actual tests.

Now that the time has arrived that chemical laboratories can have ozonizers of convenient size, and in case of successful results, can have large ozonizers of any size, it is to be hoped that earnest investigators will set to work in their particular lines and will contribute to the development of the ozone industry.

No antimony was produced in the United States last year from domestic ore, but a considerable quantity was saved in the form of antimonial lead, which is obtained at the smelters of precious metals in the course of their operations, and large quantities of antimonial alloys are recovered from secondary sources, such as scrap bearing and type metals, solder and antimonial lead drosses. From antimonial lead of both domestic and foreign origin, but smelted in the United States, according to figures compiled by the United States Geological Survey, 1949 short tons of antimony were produced, while from old alloys, scrap, dross, etc., 2506 short tons of antimony were recovered.

The Yampa smelter in Bingham canon, Utah, has been offered for sale. It was built in 1904 and for several years treated low-grade copper ores from the vicinity of Bingham. Later the company made an ore contract with the A. S. & R. Co. and the plant was closed, never to be reopened.

Gogo juice is used by the natives in the Philippines in panning gold-bearing sands. The juice is obtained from the bark of a plant and is neither alkaline nor acid. It "sickens" mercury and destroys its amalgamating power; but it is supposed to cause fine gold to settle quickly in a pan and not float away. According to experiments outlined in the Philippine *Journal of Science*, gogo juice has no qualities which would be beneficial in the concentration of gold, but the effect of a spray of the juice on the surface of water in a pan is to cause floating particles of gold to sink. Water sprayed on the pan would have the same effect.

A New Electric Steel Casting Plant

A Stassano Furnace Installation in California

By Ernst M. Schmelz

The first electric furnace of the pure Stassano type in the United States has just been installed by the Warman Steel Castings Company in their foundry at Redondo, Cal., and is

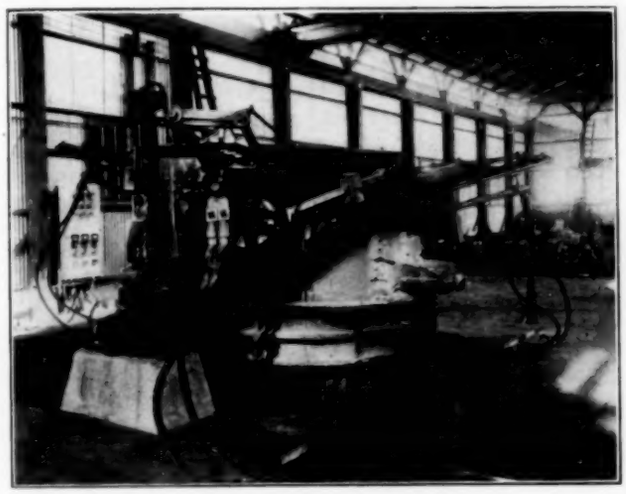


FIG. 1.—ELECTRIC STASSANO STEEL FURNACE

now in daily operation producing small and medium-sized steel castings.

This firm, which was organized three years ago, has hitherto been using Milwaukee-type crucible furnaces with a capacity of 3 tons per day. While the comparatively low price of fuel oil in California would seem to favor this process, this advantage is more than off-set by the higher cost of crucibles and labor.

One of the principle considerations in changing to the electric furnace was the fact that the firm was anxious to extend their trade to produce castings weighing 1600 lb. which is almost impracticable in a small crucible plant. An extended investigation of the various types of electric furnaces, with their particular requirements in view, decided the management on the Stassano type. An important feature influencing their decision was the adaptability of the Stassano 1-ton type for three-phase alternating current, doing away with the expense of a rotary transformer, the use of which is unavoidable if three-phase currents are used with a single-phase furnace.

The distinguishing feature of the Stassano furnace is that the metal is fused by the radiant heat from an electric arc over the metal and in a closed chamber providing a neutral atmosphere for the chemical reactions which take place in the process of refining. This permits of greater accuracy and thoroughness in closely following specified analyses in the production of castings.

The furnace in Redondo, shown in Fig. 1, is built according to the latest Stassano patents. It combines the well-known features of a completely closed melting chamber with a neutral atmosphere, the arrangement of a three-phase electric arc above the metal, preventing the carbon and all other electrode impurities from entering the metal, and allows a thorough mixing of the bath during any desired length of time.

This mixing is accomplished in a way entirely different from the older types of Stassano furnaces. Fig. 2 will explain the construction. A pair of bevel gears placed underneath the furnace and driven by a small electric motor causes the rotation of a shaft which is affixed to the center of the furnace bottom and thereby keeps the axis of the furnace always inclined at an angle to the perpendicular. The furnace is suspended by two trunnions in a horizontal steel ring, and this ring, in turn, swings by means of another pair of trunnions from supports in the same plane, but 90 deg. around from the first pair of trunnions.

This double suspension, which is the same as is used in a ship's compass, transforms the rotating movement of the furnace bottom into an oscillating movement of all furnace parts above the plane of the trunnions. In this way the water connections for the cooling of the electrodes and for the electrode hydraulic control, as well as the cable connections between switchboard and furnace, are extremely simple, while the old type of rotating furnace was rather complicated in this respect.

The furnace has a capacity of 2000 lb. to a charge and can be tilted to pour either the entire charge at once into a bull ladle or as required into small handle ladles for the pouring of small and thin-walled castings.

The electric current is supplied by the Southern California Edison Company, in the form of three-phase alternating currents at 10,000 volts and 60 cycles, and will eventually be brought down to 100 volts and 150 volts respectively by three single-phase transformers, arranged in the so-called interconnected star. Temporarily, however, until these transformers are ready, two sets of three single-phase transformers are installed, each having a ratio of 10,000 volts to 2200 volts and 2200 volts to 110 volts respectively.

As raw materials are used the mis-run castings, risers and gates, which accumulated in great quantities from the crucible process, and a small percentage of shearings, etc., supplied by local machine shops.

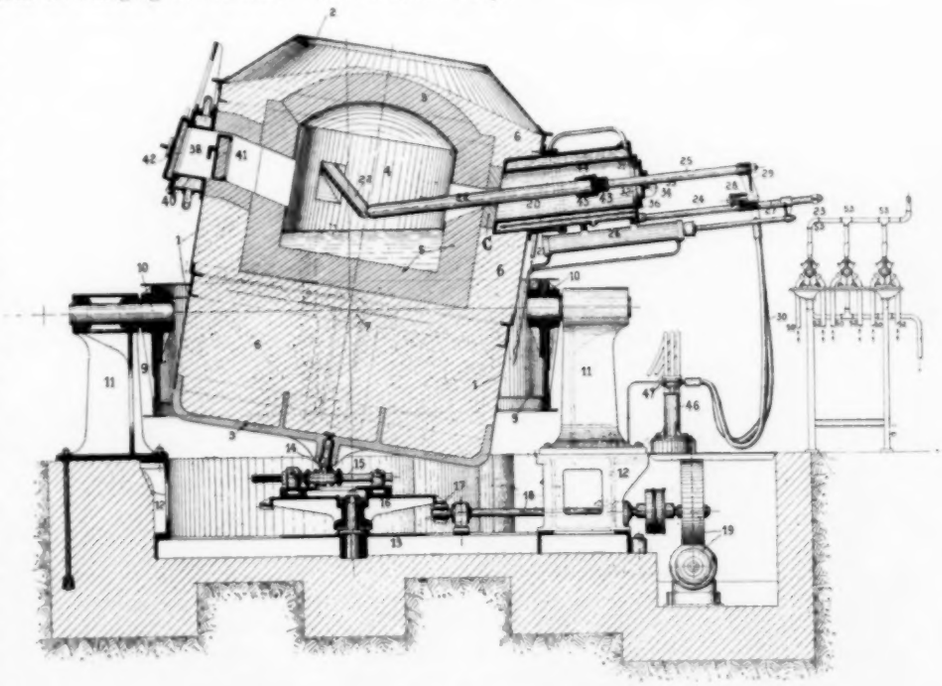


FIG. 2.—VERTICAL SECTION OF STASSANO FURNACE

The first heat from the furnace was taken on Oct. 27, and showed remarkably good results. The wattmeter installed by the power company was unfit for the control of the operations, its smallest unit being 600 kwh, but there is a recording wattmeter in the line which keeps an exact record of the operation.

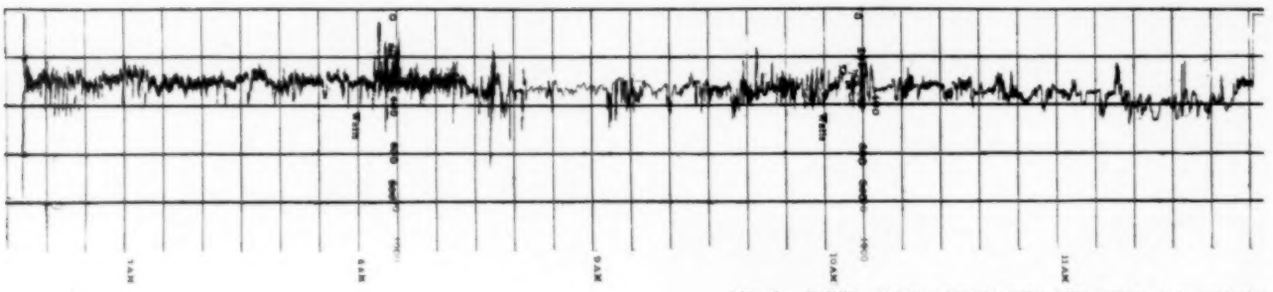


FIG. 3.—POWER CURVE DURING THE TWO FIRST CONTINUOUS

The diagrams of two consecutive heats taken off on Oct. 27 are reproduced in Fig. 3. They show, in the first place, how steady the demand of the furnace is. In spite of the fact that during the whole time the regulation of the furnace was operated by unskilled men, who until the day before had never seen an electric furnace, the demand varied hardly more than 10 per cent above or below the average.

The current consumption per ton, taking into consideration the transformer loss, figures 964 kw-hours and 1009 kw-hours per ton of charge respectively, and the time for each heat starting from a cold charge was 5 hours and 30 minutes and 5 hours and 1 minute respectively. Considering the lack of skill on the part of the men, and the fact that the current-saving arrangement of the new transformers was not yet installed, this record must be called a very good one. The writer hopes in the near future to report a further reduction, both in time and current consumption at this plant.

The metal, which proved to be extremely hot, poured to the last ladle in small castings for automobile parts. Neither a chemical analysis nor a test of physical properties has been made at the time of this writing, but the fractures at the gates showed a good grain and the castings appeared to be sound. Unfortunately the writer was unable to make further tests, being called away for the completion of another plant of the same kind in an Eastern foundry, which is expected to start operation before the end of the year.

Los Angeles, Cal.

Cost of Transportation to South America

Foreigners contemplating the purchase of American goods want to know the cost of the article delivered, and the inability or unwillingness of American exporters to quote c. i. f. prices has lost them many orders. To assist exporters in ascertaining delivery charges, the Bureau of Foreign and Domestic Commerce has published a bulletin compiled by Commercial Agent F. J. Sheridan dealing with transportation rates to the west coast of South America, this section of the foreign field being selected because of the trade activity that is expected to follow the opening of the Panama Canal. The main part of the bulletin is a series of tables giving the freight rates on over 800 articles from inland points in the United States to inland points in Ecuador, Peru and Chile. Freight rates per 100 pounds, in carload and less than carload lots, are given to New York from Chicago, St. Louis, Cincinnati, Indianapolis, Cleveland, Detroit, Buffalo, Pittsburgh, Boston, Providence and Hartford; ocean freight rates per 100 pounds and per cubic foot from New York to Guayaquil, Callao, Mollendo, Antofagasta and Valparaiso, via the Straits of Magellan and via Panama, and freight rates per 100 pounds inland from Guayaquil to Quito, Callao to Lima, Mollendo to Arequipa and Valparaiso to Santiago. Lighterage, transfer and other charges at the port of New York and at South American ports are shown, together with the cost of marine insurance, and data are also given as to consular fees and regulations and steamship requirements. Comparative tables give the competitive rates from European ports and from New York to South America. In addition to these features, statistics are furnished to show the character of the goods sold by the United States to Ecuador, Chile, Peru and Bolivia and for each of these countries general information is given as to area, population, language, currency, postage rates, foreign trade and distance from New York. Tables of price comparisons give the equivalents in the currency and measures of these countries of prices stated in the money and measures of the United States. The bulletin, in short, is a compendium of practical information for the assistance of merchants who are engaged in the export trade or who contemplate entering the foreign field. Copies of the bulletin (Special Agents' Series No. 72) may be obtained from the Superintendent of Documents, Washington, for 10 cents each.

A New Type of Electrical Furnace for the Reduction of Ores

By Francis Louvrier

In view of the many well known advantages of electricity as a source of heat, the slow development of electrometallurgy is rather surprising. Of course, numerous obstacles had to be surmounted in this development.

Most electric furnaces in use at the present time can use only

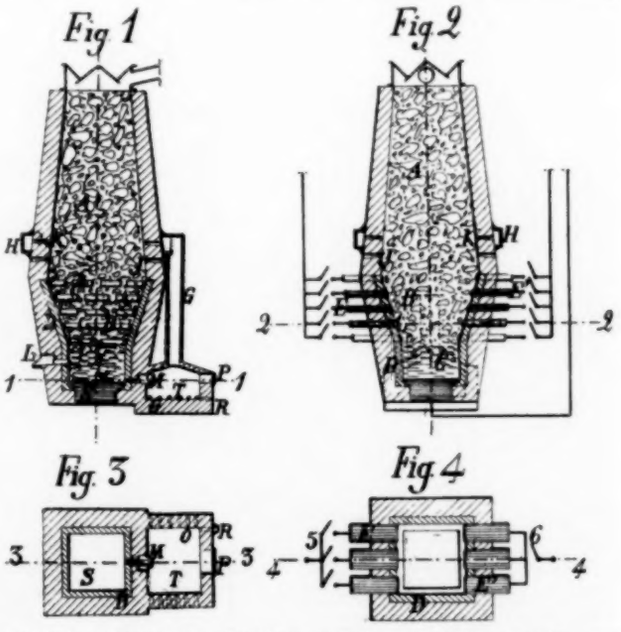
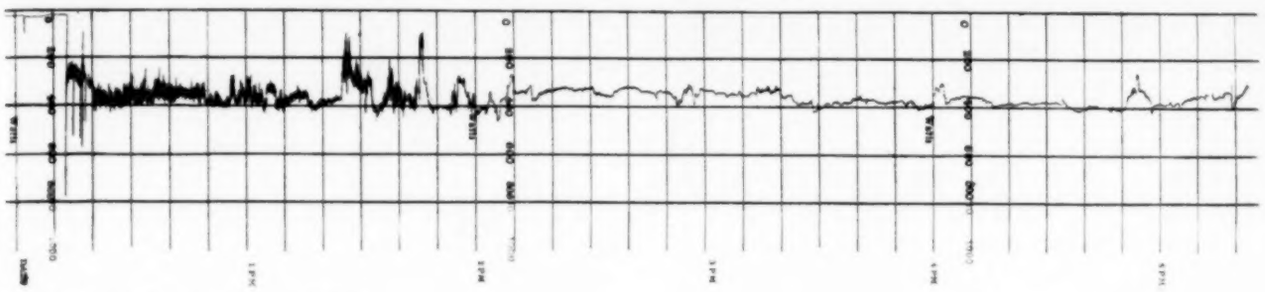


PLATE I.—SECTIONS OF FURNACE DESIGNS FOR IRON, COPPER, AND LEAD ORES

a small power, and their output is limited mainly on account of the shape of the furnace and the arrangement of the electrodes.

Temperature regulation is difficult, if carried out, by means of movable electrodes, as these are heavy and usually immersed in the charge. While it may not be easy, yet it is always possible to remove the electrodes further away from each other, so as to lower the temperature in the furnace. But the reverse operation—bringing the electrodes nearer together in order to increase the temperature—may be impossible because the charge is in the way.

In certain furnaces the electrodes are put on a slant and have to bear the weight of the charge. There is the danger



RUNS OF THE ELECTRIC STEEL FURNACE

of breaking at any moment. Further, the thermic efficiency of many electric furnaces actually in use is unsatisfactory: their consumption in electrode material is too high, and so on. Difficulties of this kind have held back the development of electrometallurgy. To overcome them has been the purpose of the furnace design which is the subject of this article.

This type of furnace, which is the invention of F. Louvrier and C. Louis, is characterized by the use of electrodes which are practically fixed, and the regulation of temperature is accomplished by simple manipulation of switches, by means of which the number of electrodes in circuit or the position which they occupy toward each other can be varied.

The general arrangement of the furnace is indicated by Plate I, which represents a polyphase furnace.

Fig. 1 is a vertical section along 3-3 of Fig. 3. Fig. 2 is another vertical section along 4-4 of Fig. 4. Fig. 3 is a horizontal section along 1-1 of Fig. 1. Fig. 4 is a horizontal section along 2-2 of Fig. 2.

The furnace proper consists of three principal parts—the shaft A, wherein the progressive heating of the ore takes place; the bosh B, which is the zone of reduction, and the crucible C, where the molten metal accumulates. The crucible and the reduction chambers are lined with very refractory and non-conducting material D. The bottom S of the furnace is conducting and is connected with one of the phases of the current. It is made from varying material to suit each particular case: for instance, for iron ore reduction it is made of a piece of cast iron, with some cooling arrangement.

A series of openings are horizontally disposed at various heights in the two opposite walls of the bosh. Through each of these openings passes a well-fitting electrode. These electrodes form two symmetric series, opposite to each other. Each of the series corresponds to a phase of the current and each electrode in the series can be connected with its corresponding phase by means of a switch.

In smaller furnaces each electrode takes up the whole width of the furnace; in larger furnaces two or three electrodes are arranged at the same horizontal height side by side, as shown in Fig. 4, so that the size of each electrode remains within reasonable limits.

The electrodes of each series are insulated electrically from the other series by the furnace wall, which is non-conducting. The electrodes are pushed through the openings into the furnace so that they touch the charge without being immersed in it. The small space between the furnace wall and the electrode is luted in order to avoid leakage of the gas formed inside the furnace.

The top of the furnace is provided with the charging arrangement, a safety valve and the necessary pipes for the removal or recovery of the furnace gases.

When the carbon monoxide which develops from the reduction of the ore is to be burned in the furnace itself, the hot air necessary for the combustion is blown in through the tuyeres J.

The electric current passes through the charge between the two series of lateral electrodes and between these and the bottom electrode. The charge acts as resistor, and as all parts of the charge are traversed by the electric current, a practically uniform temperature is obtained.

Evidently the current passing through the charge and consequently the temperature in the furnace depend on the number of lateral electrodes that are connected to the circuit and on the relative position which they occupy.

The temperature is always easily regulated by a simple movement of the electrode switches. The electrodes themselves are, therefore, immovable or nearly so; all that is necessary is to push them in from time to time in order to compensate for their consumption. As air cannot enter the furnace and as no hot gases circulate around the electrodes, this consumption is relatively small and the compensation need only take place once or twice in a week.

The crucible C is provided with two tap holes, one, L, for the slag and the other, M, for the metal.

For operation with single phase or continuous current the furnace is arranged in the same way. One possible arrangement is a conducting sole connected to one pole, and a series of lateral electrodes occupying two opposing walls of the furnace and all being connected to the other pole of the circuit. Another possible arrangement uses two lateral opposite series of electrodes connected to the two opposite poles of the circuit, without a conducting sole.

In plate I, Fig. 3, a refining crucible T with accessories (tuyeres O, chimney G, etc.) is shown; this is intended for the refining of copper.

This furnace has the following advantages: Simplicity of construction; ease of regulation of temperature by a single movement of the electrode switch; the possibility of using a high voltage, such as 120 volts in small furnaces, 240 volts in medium-sized furnaces, and 350 to 400 volts in large furnaces, according to the ore under treatment. These are the usual supply voltages, so that the electric furnace may be connected directly to the lighting and power circuit. These furnaces can be built in large sizes, even above 40,000 hp, according to the inventors. For instance, a large three-phase furnace may have a reduction chamber B 5 meters (16 ft.) wide on top and 3 meters (10 ft.) at the bottom. There are above each other five horizontal series each of three electrodes, with dimensions 0.25 meter and 1.10 meter (0.8 and 3.6 ft.). The section of the electrodes per phase is therefore $25 \times 110 \times 5 \times 3 = 41{,}250$ sq. cm., which allows of the passage of a current of at least 50,000 amp (if we allow 1.20 amp per square centimeter).

If in such a three-phase furnace the voltage between the phases is 400 volts, the power is $50{,}000 \times 400 \times 1.732 \times 0.025$ watts or 43,000 hp, the power factor of the furnace being assumed to be 0.925.

A similar furnace operated with single-phase current would have a capacity of 50,000 hp and with continuous current it would exceed 54,000 hp.

A furnace of 40,000 hp could produce, according to the inventors, 500 to 550 tons of pig per day; it could handle 2200 to 2800 tons of copper ore, or 600 tons of 50 per cent zinc concentrate.

Another advantage of the design is that the furnace is completely enclosed so that no gases can escape nor is the outside air able to enter.

The consumption of electrode material is considerably reduced since the hot gases do not circulate around the elec-

trodes and these latter can only be used up at their ends. With the exception of the ends the electrodes are out of reach of the high heat and of combustible gases, air and oxygen.

Considerable reduction in manual labor results from the large capacity of the furnace and from the facility of regulation; on account of the small consumption of the electrodes these have only to be readjusted in their positions once or twice a week.

Heat losses are avoided, except the loss by radiation, and this is small on account of the large size of the furnace.

The heat carried upward by the gases is utilized in the upper portion of the furnace for preheating the charge.

The gases, as far as they are combustible, can be burned in the furnace itself and their inherent heat may thus be utilized. Or it is possible and easy to recover completely the gases in case this is desired.

On account of the advantages just mentioned the efficiency of the furnace is high.

A uniform temperature is obtained in the whole part *B* of the furnace so that the reduction zone comprises the whole of part *B* and is not limited to a small space between two or more electrodes of small cross-section.

A uniformly progressive rise in temperature is obtained from the top of the furnace toward the lower part of the reduction zone.

The operation of the furnace is absolutely continuous, without interruption for the replacement of electrodes.

REDUCTION OF IRON ORES AND PRODUCTION OF PIG IRON OR OF CRUDE STEEL

For the reduction of iron ores the furnace is used as shown in plate 1, with or without supplementary crucible. This crucible can be installed where the direct production of cast iron or crude steel is intended.

The upper part of the furnace may be provided with tuyeres *J* for supplying the hot air necessary for the combustion of the carbon monoxide which rises upward from the reduction zone in the furnace.

The consumption of electrical energy in a Louvrier-Louis furnace which treats 50 per cent ore should not exceed ¼-hp year per ton of pig iron produced, and would be reduced to 1/5-hp year when the carbon monoxide is burned inside the furnace.

The expense for electrodes should not exceed 8 cents per ton of pig produced.

The advantages may thus be summed up:

1. Possibility of transforming smelting furnaces in actual use into electric furnaces.

2. Output as large as in blast furnaces; a large electric furnace can easily produce 600 tons of pig iron per day.

3. Better quality of product, since the electrical energy does not introduce any impurity (sulphur, phosphorus, etc.), which is always contained in the coke.

4. Possibility of directly producing crude steel, by having the molten metal during its reduction in contact with a smaller amount of carbon, and by employng pure ores.

5. Reduction of treatment cost, electrical energy, electrodes, reducing carbon, flux, manual labor, general expense, upkeep, amortization, etc., which, as estimated by the inventors, can be reduced to $5.20 per ton of pig in countries where cheap water power is abundant and where reducing carbon can be obtained at $7.50 per ton.

6. Possibility of using the heat of the carbon monoxide which rises during the reduction of the ore.

Before adopting the furnace just described, Messrs. Louvrier and Louis had patented and recommended another type of furnace which was characterized by its movable vertical electrodes between the charging stacks. This type of furnace was offered by its inventors in 1908 to the Noble Electric Steel Company for its Heroult plant at Pitt, Shasta County, California.

This now ancient Louvrier-Louis type is in actual use at the Heroult plant at Pitt, with some slight modifications in the shunting of the current, which were introduced by Mr. R. E. Frickey (see *Metallurgical and Chemical Engineering*, July, 1913).

This type of furnace with charging stacks has been abandoned by its inventors since 1910 and has been replaced by the furnace above described, which gives much superior results.

REDUCTION OF COPPER ORES

The general arrangement of the furnace is exactly as shown in plate 1. The lower part of the stack *A* is fitted with tuyeres *J* which supply hot air which is required for the oxidation of the ore and the combustion of the sulphur.

A crucible *T* serves for the refining of the matte. This crucible, which is built close to the furnace, communicates with it by an ordinary tap hole *M*, which can be opened or closed from the outside. This crucible is provided with a door *P* made from refractory material. This door also serves for introducing into the crucible the materials (silica, etc.) which are required for refining the matte.

Tuyeres *O* are arranged in the lower part of the crucible for blowing in hot air into the center of the matte.

The hot gases resulting from the refining can be led off into the upper part of the furnace by the chimney *G* into the annular space *H* and through the openings *K* in order to utilize their heat.

The crucible works intermittently while the furnace operation is continuous. It is also possible to use two crucibles as described and placed symmetrically one on each side of the furnace.

The consumption of electrical energy per ton of ore (with 5 to 6 per cent of copper) should be between 0.06 to 0.03-hp year, according to the nature of the ore and its sulphur contents. The cost of electrodes should amount to about 2 cents per ton of ore treated. The inventors claim the possibility of treating in one furnace 2000 to 2500 tons of medium-grade ore per day with a considerable reduction of operating costs. Including general expense and amortization these are estimated by the inventors to amount to $1.20 per ton of ore treated in the case in which the smelter produces its own electrical energy from water power.

TREATMENT OF LEAD ORES

The arrangement of the furnace is almost the same as described for copper ores. We will, therefore, refrain from further description.

REDUCTION OF ZINC ORES

Plate II shows the arrangement of the furnace in this case. The interior of this furnace is divided into two compartments by a horizontal refractory partition with a central opening for the passage of the ore. The lower chamber forms the actual furnace, in which the reduction of the ore takes place, while the upper chamber serves for the drying and the preheating of the charge. The partition also serves to economize space and facilitates the withdrawal through the lateral openings *O* of zinc vapors and of the carbon monoxide produced in the furnace.

These openings *O* lead into side chambers *R* and *R*, built onto the walls of the furnace (Figs. 2 and 3 of plate II). In these chambers the vapors of zinc are separated from the carbon monoxide before they pass into the condensers. The separation takes place by gravity only: the heavy zinc vapors sink to the bottom while the carbon monoxide goes to the top and passes into the upper portion of the interior of the furnace.

The use of these side chambers is further intended for the lowering of the temperature of the zinc vapors to a degree intermediate between the temperature of their production inside of the furnace and between the liquefaction in the condenser.

In order to effectively separate the carbon monoxide gas and the zinc vapor it is necessary that the quantity of gases and vapors entering is equal to the quantities escaping at any time.

For this purpose a valve *V* regulates the escape of the zinc vapors to the condensers through the tubes *H* and *K* (Fig. 2) and the adjustment of the valve is based on the indications obtained through the two taps *X* and *Z*, which are placed at

convenient heights in the lateral chambers. When the furnace works normally, the upper tap *X* must give carbon monoxide on opening; the other tap *Z* must show zinc vapors.

When the furnace works normally the proper adjustment of the valve *V* is easily found by experiment, for the production of gas and zinc vapors inside the furnace remains fairly constant.

The temperature of the condensers is uniformly maintained between 600 and 700 deg. C. The consumption of electrical energy per ton of 50 per cent zinc ore amounts to 0.2-hp year.

The expense for electrodes amounts to about 8 cents per ton of ore.

These advantages are claimed by the inventors:

Zinc is obtained as molten metal and not as "blue powder"—the proportion of blue powder does not exceed 3 per cent of the metal.

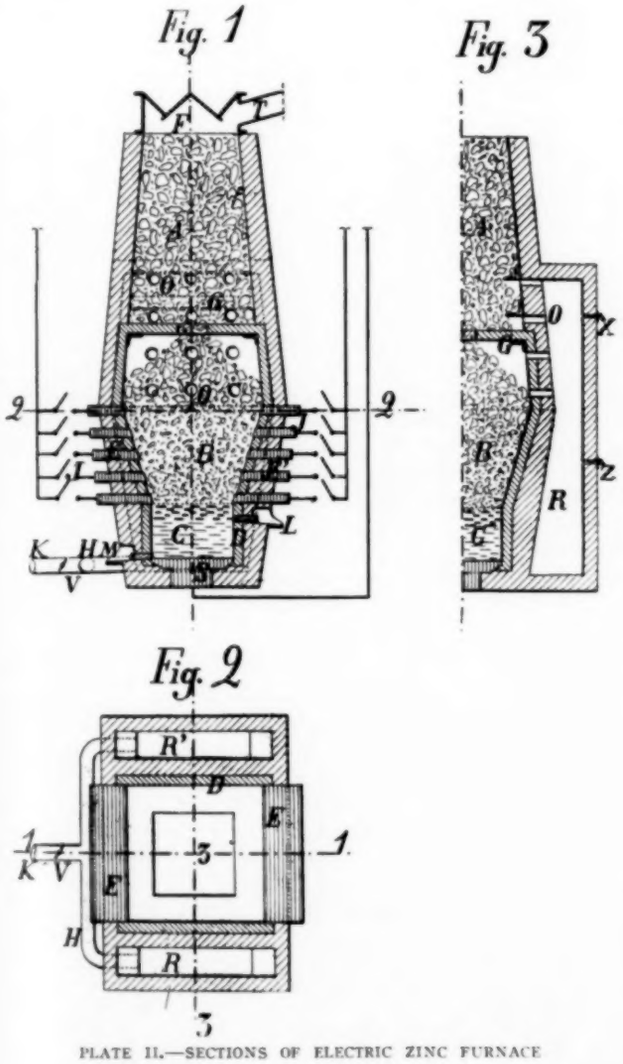


PLATE II.—SECTIONS OF ELECTRIC ZINC FURNACE

The better condensation of the zinc vapors is facilitated by their separation from the gases of reduction and by the previous lowering in temperature in the side chambers.

Most of the faults are suppressed which cause losses in metal so that a yield of at least 96 per cent of the zinc contents of the ore is obtained.

The chemical reactions are not disturbed by foreign factors, like air, moisture, etc., which may be contained in the charge, as these are completely removed before the ore arrives in the reduction zone.

The operation is continuous and adapted to large-scale working—600 tons can be treated per day by a single furnace.

Further advantages claimed are improved thermal efficiency, reduction in cost of handling, diminution in the amount of reducing carbon and the possibility of roasting the ore without expense for fuel, by utilizing in the upper part of the furnace the heat developed from the combustion of carbon monoxide which rises upward from the reduction zone.

These advantages result in a reduced operating cost. The total cost of treatment, including general expense and amortization is estimated by the inventors as $5.25 per ton of ore treated in case the electricity is generated by means of water power.

In a paper presented at the recent meeting of the American Electrochemical Society in Denver Messrs. Dorsey A. Lyon and Robert M. Keeney arrived at the following conclusions: "The chemical conditions leading to the formation of blue powder are due largely to the speed of reduction of zinc oxide in the electric furnace and to the presence of volatile impurities in the charge. . . .

"In the electric furnace, the temperature directly beneath the electrode in either the arc or the resistance furnace is so high that the reduction proceeds so rapidly that the large amount of carbon dioxide formed does not have a chance to be reduced to monoxide by the hot coal of the charge before entering the condenser.

"Hence there is oxidation of the zinc.

"By the use of electrodes of large cross-section it is probable that this concentrated heating could be reduced so that, rather than a very high temperature at one place there would be a uniform comparatively low temperature all over the charge. . . . Or the current could possibly be distributed more evenly by flat electrodes of rectangular cross section."

It can easily be seen that the Louvrier-Louis furnace fulfills all the conditions mentioned by Messrs. Lyon and Keeney.

In a paper presented at the same meeting Mr. Woolsey McA. Johnson stated that the electric reduction of zinc ore would be solved if the following conditions could be properly fulfilled:

"1. A first-rate preheater, mechanically strong and machine operated. Preheating *per se* has no advantage save to reduce kw-hours per ton of ore. Preheating must be done right, not too hot and not too cold, with a progressive application of metallurgical means.

"2. The flow of heat through the charge in the electric smelting furnace and slag must be under perfect control. The electricity must be controlled and care must be taken that the zinc globules in eductor assume no statical charge otherwise coalescence of the sphericles is hindered.

"3. A regular flow of pure zinc gas must be delivered at proper temperature to the condenser.

"4. The condenser must be durable and under good metallurgical control."

Finally, Mr. Peter A. Peterson at the same meeting summed up his conclusions on this important problem in the following manner:

"It is necessary to have the charge absolutely dry. The presence of water even in small quantities is serious, due to the oxidizing of the zinc vapor by decomposition of the water. What is still more serious is the instantaneous generation of steam and other gases, causing a great increase in the velocity of gases, thus disturbing the condensation of the zinc in the condensers and thereby causing a loss of zinc and also a loss of silver and other metals, due to furnace charge being blown outward into the condenser.

"If it is not possible to mix the charge with hot calcines from the roasting furnace and then immediately feed it to the electric furnace, there must be a preheating furnace to heat the charge before introducing it into the electric zinc furnace."

All, or nearly all, of the conditions mentioned by Messrs. Lyon and Kenney, Johnson and Peterson are practically fulfilled in the Louvrier-Louis furnace.

Most of these conditions have already been mentioned by the writer in the article entitled "Causes of the Practical Nonsuccess of Electric Furnaces in Treating Zinc Ores," which appeared in *Metallurgical and Chemical Engineering* of November, 1912.

Notes on Chemistry and Metallurgy in Great Britain

(By our Special Correspondent)

The New Headquarters of the Institution of Civil Engineers

The new building of the Institution of Civil Engineers has been almost completed. It is a very elegant structure, with a frontage of about 170 feet to Great George street, and 140 feet to Princes street when the whole design shall have been carried out.

The building is constructed of steel framework with an exterior of Portland stone, with the main entrance, which is ornamented with some very fine sculpture, in Great George street. The north and south reading-rooms are on the ground floor, and the latter apartment gives access to the council chamber and committee room; and a small hall also occupies part of the western side, while the eastern side is devoted principally to offices.

The main library is on the front of the first floor, has a length of 147 feet and a width of 28 feet 6 inches, and the old oak fittings and panelling from the building recently demolished have been used as far as possible, whilst the new panelling required, in consequence of the larger size of the new library, has been made to correspond in style. On the second floor is a similar upper library.

The chief apartment of the building is the great hall, which has a length of 100 feet and is also 45 feet in height on the Princes street side. Its mural decorations are in marble, and the ceiling is finely decorated. Three large hinged panels divide the hall from the lecture theater, which is almost exactly on the lines of the theater of the old building, and is of the same dimensions. The old oak dado was removed and has been utilized in the new theater which very closely resembles, even to the dome lighting, the theater of the old building so familiar to engineers from all parts of the world.

Electric lighting is used throughout the building. The heating is mainly by steam radiators, and adequate ventilation has been provided everywhere.

Oil-Shale in Skye

From details recently published by the Geological Survey it appears that the shale discovered in the Island of Skye is of considerable extent and reaches from fully five miles north of Portree to about the same distance south of that place. How far the seam extends westwardly cannot be ascertained at present, but for a good way inland it is at a workable depth and the dip is slight. No outcrops have been located inland, and the deposit is below strata of other rocks. The outcrop south of Portree shows evidence of the action of heat in most places; but to the north more than a mile and a half of the seam has been found free from this defect.

The thickness of the shale-seam at the outcrops varies from 7 feet to 10 feet. Analysis of the shale from the coast of Skye gave 12.8 gallons of crude oil and 7.4 lb. of ammonium sulphate per ton; whilst a sample from the Island of Ramsay—where the first discovery of the shale was made—give 12 gallons of crude oil and 6.2 lb. of ammonium sulphate per ton.

The chemist who made the analyses found the specimens to be so excessively weathered that he did not expect more than a very small yield, and the true value of the shale cannot be stated until borings are made to procure unweathered samples for analysis. It is certain that new samples from the bores would show a great improvement on the figures given by the outcrop specimens; but the profitable working of the seam depends on its being rich enough to compensate for the increased cost of coal, chemicals and transport which the situation of the deposit must entail.

The Supply of Nitric Acid in England

It is now several years since some of our leading chemists warned us that England should be made independent of foreign sources for the production of nitric acid and nitrates, both for fertilizing purposes and for the manufacture of explosives in war time; and at length there is a prospect that this country will soon be in a position to manufacture all the nitric acid which will be required.

Steps are being taken for the erection of works at Dagenham, on the Essex shore of the Thames, which will be capable of producing 12,000 tons of nitric acid a year from cyanamide, and to establish a plant with a similar capacity at Trafford Park, Manchester; in addition to which it is proposed to put down plant in Scotland to turn out 9000 tons of nitric acid a year, and in Ireland for an output of 3000 or 4000 tons. The production of cyanamide will be such as will permit heavy reserves being kept in store, but in case of necessity any waste products such as ammoniacal liquors could be utilized. It is understood that the whole of these works will be controlled by the Nitrogen Products & Carbide, Ltd.

The British Aluminium Company

The British Aluminium Company has entered into an agreement with the Burgh authorities of Burntisland to acquire some 40 acres of ground for the establishment of new works close to the town on a site outside the Borough boundary and near the road to Aberdour, and also for the possession of further ground for waste products and ulterior developments. The new factory will be in proximity to the main line of the North British Railway Company and easily accessible from Burntisland Harbor. It is estimated that the new works and plant, which are to be completed within a year and a half, will involve an outlay of about £200,000, and that from 300 to 400 hands will be employed.

Engineering Imports and Exports

The returns issued by the Board of Trade show that for the nine months ended on the 30th September there were fair increases in the values of imports and exports of engineering materials and products as compared with the corresponding period of last year, but the statistics for the month of September are not so satisfactory, and exhibit a general decline in exports except for machinery and new ships. For the nine months the imports of iron and steel, including manufactures, amounted to £11,192,013, an increase of £1,959,693; and exports totalled £40,962,292, an increase of £6,413,021. Other metals, including manufactures, were imported to the value of £24,193,890, an increase of £1,962,947; and were exported to the value of £9,972,750, an increase of £1,288,405. Imports of machinery are put at £5,444,611, a rise of £448,782; and exports went up to £27,489,564, an improvement of £3,433,179. New ships reached £27,031 for imports, and £9,139,809 for exports, showing increases of £1,723 and £4,706,767 respectively.

Taking the figures for the month of September alone, imports of iron and steel increased £21,445, but exports decreased £531,817; imports of other metals rose £284,733, whilst exports fell £162,765; in electrical goods and machinery the imports were £27,429 higher, and exports were £199,052 lower; machinery showed an increase of £33,265 in imports, and an increase of £172,895 in exports; the value of new ships imported was greater by £2,407, and exports improved to the extent of £100,065.

The New Engineering Laboratories at Dundee

In the middle of October the new engineering laboratories at University College, Dundee, were opened by Sir A. B. W. Kennedy. In the north block, which is two stories high, are situated the hydraulic laboratory, drawing offices, lecture rooms, the library and rooms for the contemplated architectural department. The tank which furnishes the water supply for the hydraulic laboratory has a capacity of 10,000 gallons, and is placed in the central tower 80 feet in height, which contains the appliances for the ventilation of the building. The water pressure available corresponds to heads of 65 feet, 140 feet and 600 feet. The equipment of the hydraulic laboratory includes a flume 45 feet long by 3 feet wide, an inward flow pressure turbine, a high-speed Oddie-Barclay reciprocating pump, an electrically driven centrifugal pump throwing 450 gallons per minute and a Pelton wheel.

The eastern wing provides for the boiler-house, the heat engine laboratory, workshop, physical testing laboratory and a separate room for cement testing. The engine room contains a horizontal tandem compound steam engine of 100 b.h.p. at 100 r.p.m. with a boiler pressure of 250 lb. per square inch. Steam is supplied from a Babcock & Wilcox boiler and superheater with a Green economizer evaporating about 1500 lb. of water per hour under ordinary conditions: and an electrically driven Worthington surface-condensing plant operate in connection with the engine. There are also fans for forced draft and a carbon-dioxide recorder.

The National Gas Engine Company contributed a 30 h.p. gas engine and a suction gas producer; and there is a 14-16 h.p. Argyle Petrol engine. Space is reserved for a Diesel engine and a steam turbine which will probably be provided in the near future. The equipment of the physical testing and cement rooms includes a 50-ton single-lever Buckton testing machine, an alternating stress machine, elasticity and rigidity apparatus, and a cement testing machine. The total cost of the building and equipment amounts to £17,000, toward which the Carnegie Trust for the Development of the Scottish Universities contributed £10,000.

A Rapid Method of Estimating Phosphorus in Steel

Accurate results are claimed for the process devised by Haripada Bhattacharyya (*Journ. Soc. Chem. Ind.*, 1913, 32, pp. 738-739). The steel is dissolved and the yellow precipitate of ammonium-phospho-molybdate is obtained and filtered off in the usual way, and the precipitate is washed with 1 per cent. nitric acid until quite free from iron: a 1 per cent. solution of potassium nitrate is then employed to completely wash out the nitric acid, when the filter paper with the precipitate on it is placed in a wide-mouthed 200 c.c. flask and 20 c.c. of N/10 soda solution are added. The excess of soda is estimated with N/10 hydrochloric acid, using phenol-phthalein as an indicator. The sodium hydroxide reacts with the phospho-molybdate according to the equation $(NH_4)_3PO_4, 12MoO_3 + 24NaHO = (NH_4)_3PO_4 + 12Na_2MoO_4 + 12H_2O$.

Market Prices

October, 1913.

Tin opened at £186, 10s. and fell away to £184, 15s. on the 17th rose to £185, 12s. by the 9th, dropped a little and again rose, reaching £186, 15s. on the 16th, and has since remained lower at £184, 10s. or about. Closes at £182.

Copper.—Opening at £74 the price gradually fell until the 15th, when it was at £71, 15s., it then recovered by degrees, rising to £75, 2s., 6d. on the 21st, and after a loss of 15s. again reached £75 on the 27th. It closes at £72, 15s., 0d.

Haematite opened at 66/6 and kept this price throughout the month, closing, however, 63/3.

Scotch Pig opened at 60/6 and fell away to 59/1½ on the 9th; recovered to 58/9 on the 10th and fell again to 57/7½ on the 13th; after reaching 58/- on the 22d it again fell to 57/6 and closes 56/9.

Cleveland opened 54/6, but had reached 53/7½ on the 7th and 51/7½ on the 13th, it then rose a little, but weakly, and was down to 51/10½ on the 17th, and 51/7½ on the 27th, and closes 50/9.

Lead opened at £20, 15s., 0. and fell slightly to £20, 5s. by the 8th; recovered next day to £20, 10s. and to £20, 15s. on the 15th, remaining at the latter price, but closing £21.

	£. s.d.
Aluminium, ingots, ton	96. 0.0
Alum, lump, loose, per ton	5.10.0
Antimony, Star Regulas	29. 0.0
Borax, British refined crystal, cwt.	18.6
Copper, sulphate, ton	24. 0.0
Copper ore, 10 to 25%, unit	12/9 to 13.3
Ebonite rod, lb.	4.6
Hydrochloric acid, cwt.	5.0
India-ruber, Para, fine	3.1½
Mica, in original cases, medium	3/6 to 6.0
Petroleum, Russian spot, gal.	9¼
Quicksilver (Spanish), bottle	7. 5.0
Sal ammoniac, lump, firsts, ton	44. 0.0
Sulphate of ammonia, f.o.b Liverpool, ton	13. 5.0
Sulphur recovered, ton	5.10.0
Shellac, cwt.	4.14.0
Platinum, oz., nominal	9. 5.0
Tin ore, 70%, ton	£113 to 115. 0.0
Zinc, Vielle Montagne, f.o.b. Antwerp	25.10.0

DIFFERENCES

Higher.	£. s.d.	*Lower.*	£. s.d.
Copper, ton	5.0	Tin	7.5.0
Lead, ton	10.0	Antimony, ton	1.0.0
		India-rubber, lb.	5½
		Tin ore, ton	10.0.0
		Shellac, cwt.	4.0
		Zinc, sheet	1.0.0
		Haematite, ton	4.0
		Scotch pig	3.3
		Cleveland	4.0

Recent Chemical and Metallurgical Patents

Gold and Silver

Solvent for Precious Metals.—Halogen cyanides have been found good solvents for gold and silver, and the bromocyanide process is extensively used in Australia. Care must be exercised in the preparation of bromocyanide, or bromine will be wasted. A new method of generating halogen cyanides in connection with an oxidizing agent has been patented by Dr. HANS FOERSTERLING, of the Roessler and Hasslacher Company, of Perth Amboy, N. J. The inventor melts sodium bromide and sodium cyanide in the proportion of one molecule of each, and allows the mixture to solidify. No decomposition takes place in this process. In order to convert this into bromocyanide, it is treated with an acid solution of hydrogen peroxide, and the reaction proceeds as follows:

$$NaBr + NaCN + H_2O_2 + H_2SO_4 = BrCN + Na_2SO_4 + 2H_2O.$$

If an acid was not used, sodium hydrate would form, and destroy the bromocyanide. The rapidity of formation of bromocyanide according to the above equation depends on the concentration of the solution. The formation of the corresponding chlorine and iodine cyanides according to the same formula, is much more rapid than the formation of bromocyanide.

In the practical use of this invention in ore treatment, the mill operator would receive the correctly proportioned mixture of sodium bromide and sodium cyanide, and all that would be necessary would be to dissolve the salt and add an oxidizing agent. An acid solution of hydrogen peroxide is preferred for this purpose, because it is stable when formed by dissolving sodium peroxide in an excess of acid. Three parts of sodium cyanide should be used with one part bromocyanide in order to dissolve gold, thus:

$$BrCN + 3NaCN + 2Au = 2NaAu(CN_2 + NaBr.$$

Sufficient cyanide can be added originally to the alkali bromide to give a mixture that will act in a manner equivalent to the two equations:

$$NaBr + NaCN + H_2O_2 + H_2SO_4 = BrCN + Na_2SO_4 + 2H_2O.$$

$$BrCN + 3NaCN + 2Au = 2\,NaAu(CN)_2 + NaBr.$$

The largest part of the alkali halogen is not lost in practice, and only so much need be added as to make up the small loss.

Combined Pulp Agitator and Thickener.—In the cyanide process, the steps of thickening and agitating the ore pulp in solution are carried on separately. In Fig. 1 is shown an apparatus in which it is proposed to perform the two operations within the same tank, thereby reducing the cost of apparatus and its maintenance. The device is the patented invention of Mr. J. V. N. DORR, and embodies the principles of his continuous thickener and an air-lift. A partition in the form of a truncated cone divides the tank into two compartments in which the different processes of decantation and agitation can be carried on without interfering with each other.

A flat-bottom tank A, with overflow launder H, contains a

partition E in the form of a truncated cone, dividing the tank into the compartments Y and X. Within the inner compartment is an air-lift C supplied with compressed air from pipe D. A central shaft K is operated by the mechanism ML, and carries at its lower end four radial arms J fitted with rabbles which tend to rake settled solids toward the center.

The proposed operation is as follows: The pulp is fed through launder B to an annular distributor G. The solids tend to settle, while the clear solution flows from the tank at the peripheral launder H. The settling solids are raked to the center by the arms J, where they are acted upon by the air lift, raised and discharged within the compartment X. Thus the solids, which require aeration and agitation to cause dissolution of their precious-metal content, are given the additional treatment required. A portion of the thickened and agitated pulp is

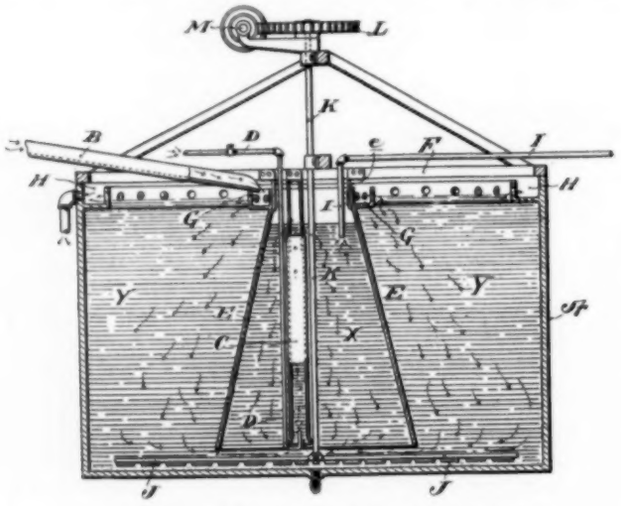


FIG. 1.—COMBINED THICKENER AND AGITATOR

constantly drawn off through the pipe I. Several modifications of the apparatus can be designed, introducing several concentric cone partitions, with a plurality of air-lifts. (1,076,666, Oct. 28, 1913.)

Copper

Apparatus for "Amphidizing" Ores.—In this journal, April, 1913, page 218, reference was made to the Bradley process for "amphidizing" copper ores, being a method of rendering copper soluble, as sulphate. In a recent patent granted to Mr. Charles S. Bradley, of New York City, the inventor gives a description of the "amphidizer" or oxidizing roaster used to convert the copper ores into soluble condition. The device is essentially a rotary kiln with interior passages so arranged that the ore passes from one end to the other and back again before it is discharged with its copper in soluble form. Heat is supplied through a central passage, and the ore coming in is heated continuously and in a regenerative manner due to the fact that the returning ore transfers some of its heat to the incoming charge. (1,077,036, Oct. 28, 1913.)

A chemical filter, specially adapted to the filtration of acid, corrosive pulps, is the basis of another patent granted to Mr. Bradley. As shown in transverse vertical section in Fig. 2, it is a flat filter bed with a foundation 10 and side walls 11 and 12 composed of concrete. The filtering medium is porous silica brick molded with suitable channels and passage ways connecting with the conduits 19, the latter being connected with sources of vacuum and pressure as desired. The pulp to be filtered is delivered into the channels 15 and 16 in the side walls, and flows onto the bed by way of the ports 18. Vacuum being applied, the solution is drawn through the brick filter into the passage ways and finally to the channels 19. The filtered pulp remains on the surface, and is removed by a traveling scraper which is set to remove all but a thin layer of the material. After this is accomplished, air may be forced through the filter bed from beneath, and the thin layer of solids blown from the pores of the filter. This becomes mixed with the next charge, and the bed is in condition for another filtering operation. (1,077,037, Oct. 28, 1913.)

Metallurgical Furnaces

Improvement in Ore Roasting Furnaces.—In that type of ore roasting furnaces used for desulphurizing blende or pyrite, having an annular rotating hearth and stationary annular arch to which rabbles are attached, it has been discovered by

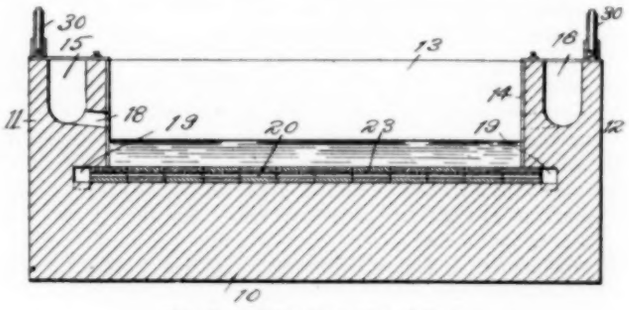


FIG. 2.—FILTER FOR ACID PULPS

Xavier de Spirlet, of Brussels, Belgium, that incipient fusion of the sulphides may occur on the hearth at the point of feeding. M. de Spirlet proposes to obviate this difficulty by arranging ventilating passages in the roof or arch of the furnace and the suspended rabbles, through which air or gas can be passed for the purpose of removing some of the heat concentrated at the point of incipient fusion. (1,077,010, Oct. 28, 1913.)

Air-Cooled Rabble Arm.—The rabble arms used in roasting furnaces of the superimposed-hearth type are usually cooled by allowing air to pass through them by way of the hollow central shaft. An improvement in the manner of cooling has been patented by Maurice van Marcke de Lummen, of Cologne, Germany, as shown in Fig. 3. The arms are made in one piece in the form of a hollow body containing a longitudinal partition *k*, which divides the interior of the arm into two compartments. The passage or compartment *l* registers with a central opening in the hollow shaft at *o*. Each passage *m* terminates at the central part of the arm in an opening that communicates with lateral passages *q* in the hollow shaft. By this arrangement the cooling air rises through the central part of the shaft, traverses the upper and lower passages of the arms, and makes its exit through the lateral passages in the shaft. (1,076,297, Oct. 21, 1913.)

Heat Regenerator.—In the use of the cheaper grades of gas fuel in metallurgical furnaces, difficulty is sometimes experienced in maintaining a sufficiently high temperature in the furnace to produce a neutral or reducing action. The object of an invention of Mr. Theodor L. Holle, of St. Louis, Mo., is to overcome this defect by preheating the air introduced into the furnace to a high temperature. This is accomplished by a plurality of circuitous passages or flues alternately arranged

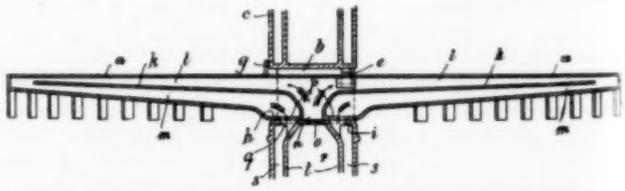


FIG. 3.—AIR-COOLED RABBLE ARM FOR ROASTING FURNACES

for the passage of air and gases of combustion in opposite directions, so that the outgoing products of combustion passing from the furnace to the stack will highly heat the flue walls, and the incoming air traveling in the opposite direction in adjacent flues will receive heat in proportion to the temperature of the walls. A relatively large number of flues are arranged in a small space, and special structural units are used. (1,076,426, Oct. 21, 1913.)

Water-Cooled Port for Open-Hearth Furnace.—In Fig. 4 is shown a longitudinal section through one end of an open-hearth furnace, equipped with water-cooling means invented by Mr. ANSON W. ALLEN, of Birmingham, Ala. The furnace is of the usual construction, having a gas port 10 and air ports 11. These are separated by an arch 12 of refractory material extending to the point where the best combustion is obtained. Beneath the refractory arch is another composed of metallic

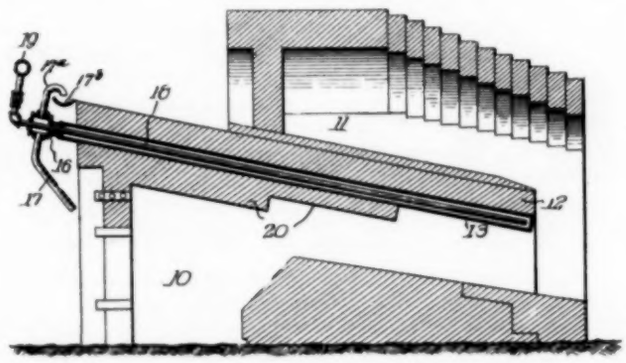


FIG. 4.—WATER-COOLED FURNACE PORT

units 13, these being trapezoidal in cross section and closed at their ends with plugs. The upper plug is apertured to receive a fitting 16, connected with an overflow pipe 17. An additional overflow pipe is provided at 17*a*, delivering into a trough 17*b*. Water is supplied separately to each unit through the header 19 and pipe 18. The cooling units are removable without delaying the operation of the furnace. (1,076,540, Oct. 21, 1913.)

Metallic Alloys

Copper-Aluminium.—An improvement in the manufacture of aluminium bronze is suggested in a patent granted to Mr. EDWARD D. GLEASON, of New York City. Boron is used in the preparation of the alloy. Into a fused bath of three parts calcium fluoride and one part fused boracic acid, ingots of aluminium are introduced. When the mass is molten it is stirred and cast into ingots. This mixture is then added in an amount slightly in excess of 10 parts to 90 parts of pure copper melted under a cover of charcoal and black oxide of manganese. The resulting alloy is free from oxide of aluminium, which is said to appear in the ordinary aluminium bronze. The alloy can be made in different degrees of hardness with the same percentage of copper by adding aluminium with more or less boron. (1,076,973, Oct. 28, 1913.)

Lead-Copper-Tin.—Plastic compositions of lead-copper and lead-copper-tin, such as are used for bearing metals, sometimes develop defects due to the lack of affinity of the lead for the copper or copper-tin, resulting in segregation or sweating of the lead. According to an invention of Mr. EDWARD D. GLEASON, such alloys can be made perfectly homogeneous by incorporating a non-metallic derivative of lead, such as the sulphide mineral galena. For example, he may melt 60 lb. copper and add thereto 5 lb. galena, and then gradually add to this mixture 30 lb. lead and 5 lb. tin. (1,077,698-699-700-701, Nov. 4, 1913.)

Malleable Zinc Alloy.—According to an invention of Mr. THOMAS A. BAYLISS, of Warwick, England, a malleable and tough alloy is produced from zinc, aluminium and lead, when the ingredients are present in proportions between the following limits:

Zinc	99.100%	and	99.9%
Aluminium	0.001	and	0.1
Lead	0.010	and	0.0

The metal thus obtained is said to be extremely tough and strong, and is malleable and suitable for stamping. (1,074,234, Sept. 30, 1913.)

Hard Copper Alloy.—The following formula gives the ingredients of an alloy patented by Mr. WM. F. NOLAN, of St. Paul, Minn.: Copper, 98.15 per cent; manganese 1.48 per cent; vanadium 0.25 per cent; aluminium 0.12 per cent. The claims for the alloy include increased durability, and a degree of hardness which makes the metal useful for cutting tools. After the alloy is mixed, and while yet at dark red heat, it is plunged into oil.

The temper can be removed by heat, but the alloy can be hardened again by heating and immersing in oil. (1,074,285, Sept. 30, 1913.)

Alloy for Metal Tape.—A lead alloy suitable for use as metal tape in winding electrical conductors is patented by Mr. CHARLES P. MCCONNELL, of New York City. The alloy contains lead, antimony and tin in the following proportions:

	Preferred	Maximum	Minimum
Lead	95.00%	96.75%	93.25%
Antimony	4.50	6.00	3.00
Tin	0.50	0.75	0.25

A satisfactory tape made from this metal need be only 0.003 in. thick and can be 1 in. wide, and at the same time have as great or greater strength than tape made entirely of lead 0.006 in. thick and 0.75 in. wide. A material saving of lead is accomplished which is in excess of the cost of the metals added to make the alloy. (1,075,661, Oct. 14, 1913.)

Manufacture of Sodium Alloys.—EDMUND CHARLES ROSSITER of Birmingham, England, alloys sodium with other metals in the nascent state. The temperature need never rise above 750 Centigrades during the whole process. To prepare an alloy of sodium and lead, for example, sodium hydrate is fused and reduced with carbon in the form of charcoal. The pot is covered and filled with dry coal gas or other suitable gas. The lead is added in liquid form before or after the reduction takes place, or the mixture of sodium hydrate and carbon is poured into the molten lead. (1,073,523, Sept. 16, 1913.)

Refractories

Refractory Material.—WALTER ARTHUR of Schenectady, N. Y., assigns to the General Electric Co. two processes of manufacturing a refractory material of valuable heat insulating quality which will withstand temperatures of from 900 to 1100 centigrades or even higher. It has a thermal resistance of from 900 to 1200 thermal ohms per inch cube, the temperature cofficient being very small. It contains mainly silicon, carbon, oxygen, small amounts of metallic impurities and aluminum in one patent, titanium in the other. The charge consists of silica, coke, and either fire clay as an example of aluminum silicate or rutile in the second case; this is subjected to the smothered arc of an electric furnace. A volatile product, representing the insulating material, condenses in the form of soft, coherent, somewhat felty masses of grayish or yellowish color; when containing manganese the color is a bluish gray. The apparent density is 0.06 to 0.16, the real density about 2.56. A typical analysis is 45.51% Si, 7.57% C., 40.13% O., 1.07% Ti., .32% Mn, 1.63% Al., 1.08% Fe. (1,072,413 and 1,072,414, Sept. 9, 1913.)

Composite Silicon Carbide Refractory Lining.—Silicon carbide is excellent as a furnace lining for its infusible and refractory nature, but has the disadvantage of high heat conductivity, being a conductor of heat five times better than firebrick. Mr. FRANK J. TONE, of the Carborundrum Company, Niagara Falls, has devised a lining in which the working portion exposed to high temperature and requiring stability and strength, is formed of silicon carbide of high refractability and low thermal resistivity, and backs this layer up with silicon carbide of a form which, although comparatively lacking in strength and refractability has very high thermal resistivity. For the first named exposed or working portion he uses bricks made of dense silicon carbide, obtained by subjecting a mixture of silicon carbide and carbon to the action of silicon-containing vapors at high temperature. For the second portion he uses porous silicon carbide, obtained by subjecting porous carbon to silicon-containing vapors at a high temperature. This composite silicon-carbide lining may be backed up by ordinary means such as the steel furnace shell or by walls of less refractory brick. (1,078,525, Nov. 11, 1913.)

Sintering

Sintering.—Messrs. Quincy Bent and Edwin Barnhart, of Sparrows Point, Md., and James B. Ladd, of Ardmore, Pa., patent improvements in the preparation of ore for sintering. The sintering process consists in passing air through the materials, after ignition, thereby causing combustion through the mass to fuse or partly fuse materials in it and to get the mass of fine material into a more or less solid mass suitable for treatment in smelting furnaces. The combustion is supported either by combustible material originally in the fine ore or by carbon purposely added. The new point is that in sintering these fine materials, which must be in a moist state, much better results are secured by separating into small parts the more or less coherent mass of material and by lightly and evenly sprinkling or depositing these small and separate parts so that the entire mass to be treated shall lie loosely and be of uniform density and consistency. (1,078,988, Nov. 18, 1913.)

Iron and Steel

Vanadium in Iron.—Mr. John Kirby, of Pittsburgh, Pa., incorporates vanadium in muck iron to increase the tensile strength by the following process. He feeds into a heat of muck iron in a puddling furnace about the time the cinder is boiled off and before the iron is ready to ball, a composition including vanadium oxide with a reducing agent and an agent to increase the combustion of the reducing agent, so that the vanadium will be reduced and thoroughly incorporated with the iron. Such a composition may contain 32 parts of vanadium oxide, 8 to 16 aluminium, 8 iron rust, 2 potassium chlorate, 1 black oxide of manganese and 1 barium dioxide. (1,079,129, Nov. 18, 1913.)

Electric Furnaces

Melting Resistor Furnace.—Mr. Albert E. Greene, of Chicago, Ill., patents a resistor furnace for melting. Claim 2 relates to "the combination with a furnace chamber, of electric resistors constituting the primary circuit for an alternating electric current flow, electric resistors arranged in inductive relation to said primary circuit and thereby constituting the secondary circuit, said electric resistors being arranged in proximity to said furnace chamber; whereby heat developed in said resistors is transformed to the charge in said furnace. (1,078,619, Nov. 18, 1913.)

Electrode Holder.—In Fig. 5 is shown a top view of a holder for electrodes used in arc furnaces. A side view is given in Fig. 6. It is designed to provide an easily adjustable support, and one which will make a good electrical contact with the electrode so that hot spots will be avoided. It is the invention of Mr. John Orr, of Schenectady, N. Y., and the patent is assigned to the General Electric Co. The holder consists of an adjustable collar consisting of a number of contact plates joined by links at the adjoining ends, and adapted to make a good contact with the electrode surface when drawn together. A bracket I forms a support for the electrode and is attached to the furnace framework. It is longitudinally adjustable by a screw 2 moving in a guide through a block 3. At the end of the bracket is a contact plate 4, which forms one part of the electrode holder. The other contact plates 5, 6 and 7 are adjustable with respect to the fixed contact plate 4, and preferably consist of some good conductor such as copper. The adjacent ends of the adjustable electrodes are connected by links 8 movably attached to the plates by pins 9. In order to give a good contact with the electrode, the contact plates have the same configuration, round or square, as the electrode. When the plates 5 and 7 are drawn toward the fixed plate 4 by tightening the screws 11, they exert pressure on the enclosed electrode and provide a good contact. (1,076,054, Oct. 21, 1913.)

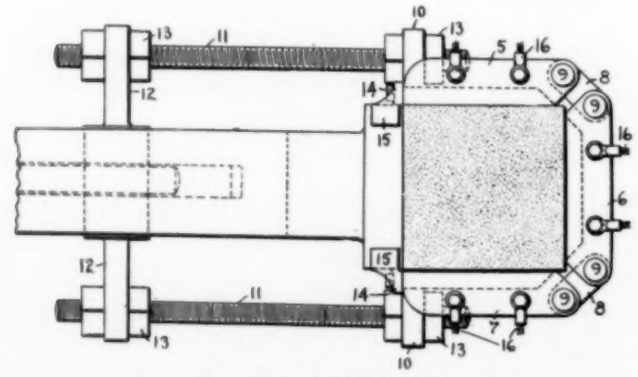


FIG. 5.—ELECTRODE HOLDER

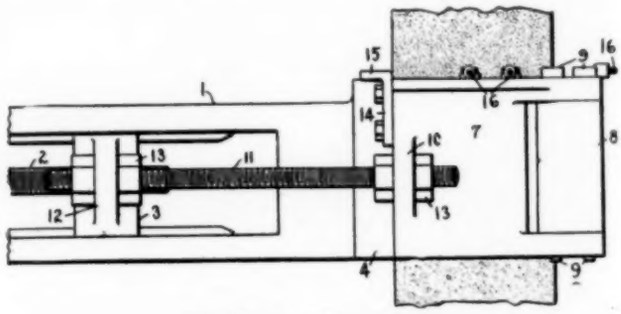


FIG. 6.—ELECTRODE HOLDER

Induction Furnace.—Leslie E. Howard, of La Grange, mettallurgist of the Simmonds Mfg. Co., describes a new electric melting furnace for bronzes, brass mixtures, white metal, Babbitt, aluminium, iron and steel. The furnace is of the induction type and is characterized by the arrangement of the secondary circuit in form of a fused metal bath of wide cross section, which forms about one-third of the secondary, while two-thirds consist of a solid mass of metal having a very large cross-section relative to the cross-section of the molten portion of the secondary. This construction allows to approach as much as possible open hearth conditions for the molten metal and further to place primary and secondary circuit very close to the magnetic field with the resulting advantage of reducing magnetic leakage. The patent is illustrated. (1,076,887, Oct. 28, 1913).

Electrolytic Furnaces

Electrolytic Furnace.—David J. Hauss designs an electric furnace adapted to produce metallic alloys or chemicals or compounds with separate regulation of the temperature at the anode and cathode. It also contains a separate chamber in which the product accumulates and is preserved from alteration. The main feature of the patent is the combined application of alternating and direct current. Fig. 7 shows the electrodes E and F (the former one installed in the special chamber) connected to a source of direct current, while the electrodes H and E supply the alternating current for heating.

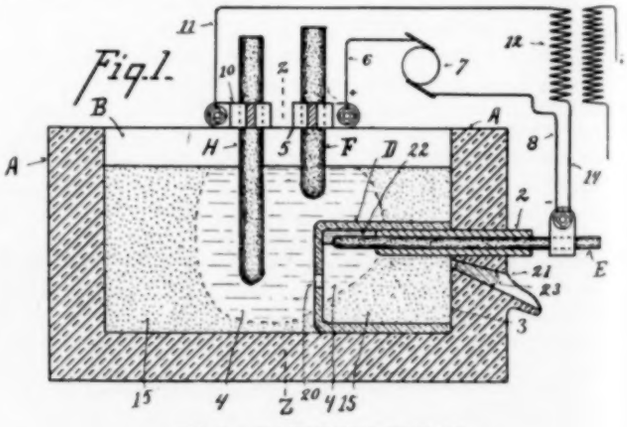


FIG. 7.—ELECTROLYTIC FURNACE

purposes. For the production of alloys the electrode F may be subdivided into two connected in parallel, one being carbon, and the other the alloying metal, a rheostat regulating the proper distribution of the current (1,060,839, May 6, 1913).

Metallic Magnesium.—For the production of metallic magnesium Messrs. ROGER WILLIAM WALLACE and EUGENE WASSMER, of London, England, electrolyze a fused mixture of magnesium chloride and magnesium sulphide in a vessel acting as one electrode with another electrode in the center. One of the advantages is that the nascent chlorine combines with the sulphide so as to form sulphur chloride which does not attack the vessel and electrode. The voltage is 8 to 10. The process is carried out in a closed vessel with a nitrogen atmosphere. (1,079,079. Nov. 18, 1913.)

Electrolytic Apparatus

Diaphragm.—A new combination of materials for an electrolytic diaphragm has been patented by Mr. HERMAN A. WAGNER, of New York City. It consists of a plate molded of a substance such as manganese dioxide, in which has been incorporated mercury and palladium black. The plate is used between porous walls of a substance such as burnt clay. In making the diaphragm, the manganese dioxide is powdered, mixed with mercuric chloride, and treated with an alkaline solution of formic aldehyde to reduce the mercury salt to metal, which is thus finely disseminated through the mass. Palladium black can then be added, and the whole molded and pressed into any desired shape. (1,077,444, Nov. 4, 1913.)

Electrode.—Metallic tungsten is insoluble in ordinary acid or neutral solutions, and can be used as a cathode in such solutions; but it is not useful as an anode because it does not readily allow the current to pass. Mr. ROYALE H. STEVENS, of Salt Lake City, Utah, has discovered that tungsten can be formed into a useful anode material by means of a very thin plating of a metal of the gold group, such as platinum, iridium, ruthenium or gold. The combination gives electrodes that are non-oxidizable and refractory, especially when used in electrolytic work where the current liberates chemically active gases at the anode. The same effect can be obtained by alloying the precious metal with the tungsten, but the plating is preferable as requiring less of the expensive metals. The plating can be accomplished with a current density of 0.05 ampere per square decimeter of the area to be plated. The electrolyte consists of 5 parts of platinum or iridium chloride, 250 parts of disodium phosphate, 50 parts ammonium phosphate, all in 1000 parts water. The solution is boiled until all free ammonia is expelled. (1,077,894-920, Nov. 4, 1913.)

Oxygen and Hydrogen.—In the electrolytic cell of RENE MORITZ, of Wasquehal, France, for the production of oxygen and hydrogen from water, the chief features are details of design by which the two gases produced are carried away to separate containers through independent and isolated ducts and any leakage from one to the other is prevented so as to insure high purity of the gases. Each electrode is bipolar, hydrogen being evolved on one side and oxygen on the other. Means for circulation of the electrolyte are provided. (13,643, Nov. 11, 1913.)

Mold for Electrotyping Processes.—In order to produce a good and lasting mold for silver plating in cyanide, ARTHUR J. TIZLEY brings strips of softened celluloid of required dimensions into intimate contact with the design to be reproduced. The celluloid mold is preferably made up of a plurality of thin sheets, and wire mesh is embedded into the celluloid as a reinforcing element. (1,063,749, June 3, 1913).

Electric Gas Reactions

Ozonizer.—NEVIL MONROE HOPKINS, of Washington, assigns to the Terpezone Co. of New York the construction of a high-voltage ozonizer. It consists of an inner metal cylinder as one electrode which is filled with a loose permeable mass of copper foil or shavings, an annular passage for the air to be treated, a glass cylinder as dielectric, bearing a heavy metal coating as the other electrode and radiators connected to this electrode for cooling purposes. The whole apparatus is enclosed in a strong outer glass tube which is perforated in one place for the introduction of a binding-post. Properly constructed caps close the two ends and provide for the in- and outlet of the cooling air and heated air. (1,062,974, May 27, 1913).

Batteries

Lelanche Cell.—For use as depolarizer in Lelanche cells, Mr. MORDUCH L. KAPLAN recommends the use of dihydrated manganese peroxide, MnO_2, $2H_2O$, produced by treating manganese carbonate with a highly diluted potassium permanganate solution at the boiling temperature of water

$$3\,MnCO_3 + 2\,KMnO_4 + 10\,H_2O = 5\,MnO_2 + 10\,H_2O + K_2CO_3 + 2\,CO_2.$$

This dihydrated manganese peroxide consists of a fine, comparatively dense powder of a grayish blue color. Cells made up with it "show 1.8 volt and are superior with respect to yields and recuperative powers." (1,078,788, Nov. 18, 1913.)

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver.

Charcoal as a Precipitant of Gold in Cyanide Solutions.—A study of the causes of precipitation of gold from cyanide solutions by means of charcoal is presented in a paper by Mr. MORRIS GREEN, in *Bulletin* No. 109, Inst. Min. & Met. (London). The author considers three possible explanations of the precipitating effect of charcoal: (1) A chemical action in which the carbon participates; (2) a reaction due to osmotic pressure; (3) a reaction due to occluded gases. The results of investigations suggest that the first two assumptions are fallacies, and that precipitation is due to the latter cause.

On the general effect of charcoal, the author presents experiments in which he added freshly ignited charcoal to different portions of the same gold-bearing cyanide solution and agitated the mixtures for varying periods. After agitation for 3, 6, 9, 12, 15 and 18 hours, the various solutions were practically identical, as also were the amounts of gold carried by the charcoal. It is concluded that either the solution is incapable of dissolving the precipitated gold, or that precipitation and dissolution proceeded at the same rates. The charcoal will not precipitate unlimited quantities of gold. Gold which had been precipitated by charcoal was not readily dissolved by fresh cyanide solutions.

The effect of different forms of charcoal was determined. The table shows the percentages of gold precipitated by equal weights of different forms of charcoal, acting under similar conditions on equal quantities of the same gold-bearing cyanide solution.

Precipitating Agent	Per cent gold precipitated
Freshly ignited wood charcoal	90.3
Wood charcoal exposed in lumps for about 6 mos.	50.0
The above charcoal reheated to incipient redness	84.2
Sugar charcoal (by heating in closed retort)	62.1
Sugar charcoal (by burning in contact with air)	5.6
Sugar charcoal (by H_2SO_4, and washing out acid)	0.0

The effect of the presence of charcoal in a cyanide solution during the dissolution of metallic gold was to precipitate some of the gold, and establish a condition of equilibrium between precipitated and dissolved metal. This equilibrium was not changed after agitation for a number of hours.

All of the foregoing results establish, in the opinion of the author, the fact that carbon *per se* does not participate in the reaction. He further shows that graphite not only does not precipitate gold from cyanide solutions, but aids solution by the formation of an electric couple where gold and graphite meet.

A number of experiments were performed to test the possi-

bility or reaction due to osmotic pressure, but with negative results.

In testing the action of occluded gases, it was found that the removal of the loosely-held gases was not sufficient to destroy the precipitating effect of the charcoal; but that when charcoal was subjected to the action of a powerful vacuum, with the addition of heat, its precipitating effect was decidedly diminished.

Nature of Charcoal

	Per cent gold precipitated
(1) Charcoal exposed to air for about two weeks after manufacture	93.3
(2) The same charcoal (1) heated and exhausted of gases	17.0
(3) Charcoal exposed to air for about one year	41.2
(4) The same charcoal (3) heated and exhausted of gases	14.8

"It appears, therefore, that the power of charcoal as a precipitant of gold from aurocyanide solution is due to occluded gases." Carbon monoxide and hydrogen were found in the exhausted gases: but since hydrogen was found in quantities too small to account for the observed effects, carbon monoxide is considered to be the active reagent.

Use of Lime in Cyaniding.—The almost universal adoption in cyanide practice of crushing ore in cyanide solution instead of water has introduced new metallurgical problems, one of which is the best method of adding lime to the ore. Mr. A. M. Merton discusses the general use of lime in cyaniding in *Mining Science*, September, 1913. He makes the rational criticism that few mill designers make adequate provision for handling lime in the mill or for the manner of its addition to the ore. The result is the ineffective use of an expensive reagent.

The functions of lime in cyaniding are classified thus:

(1) Neutralizes acids formed by the oxidation of sulphides, etc.

(2) Breaks up metallic salts, forming innocuous hydrates.

(3) Protects the cyanide solution against carbon dioxide.

(4) Furnishes alkali to break up silver minerals.

(5) Protects cyanide solution against latent cyanicides in the ore.

The effect of actions (1) and (2), and to a certain extent (5), becomes apparent during crushing and grinding. The amount of lime to be added for these purposes cannot be determined in the laboratory. The amount of protective alkalinity to be carried must be determined experimentally in the mill. In the case of Mexican silver ores it is common to have the solution at all times a concentrated lime solution.

Lime losses occur as follows:

(1) Lime-consuming constituents in the ore.

(2) Lime destroyed by CO_2 in air used in agitating.

(3) Mechanical loss in solution discharged with tailing.

The author makes a point of the fact that the mere addition of lime to the ore in quantities indicated by laboratory tests will not suffice always to protect the cyanide solution. In the case of an all-sliming treatment, for instance, although the pulp going to the agitators is alkaline, the tube mill grinding classifier sand may be working in acid solution because the lime added at the head of the system is passing from the classifier in the slime, leaving an unprotected sand to be reground. In such circuits it is necessary to see that the solution is properly alkaline at all points, and for this purpose it may be neceesary to add the lime at different points instead of at the beginning.

Lime added at the coarse crusher may be only partly effective owing to the impossibility of neutralizing the acid conditions in the larger pieces. If only a part of the lime were added here, sufficient to act on the finely crushed part, and the balance at the secondary grinder where the ore is further comminuted, a saving may be effected and greater efficiency obtained at the same time. Thus in treating an ore requiring 16 lb. lime per ton, the most effective way of adding it was as follows: At the coarse crusher, 3 lb.; at the stamps, 9 lb.; at the tube mill, 4 lb. In order to get the correct lime conditions for this ore it was necessary that the proper amount be present at each stage of the treatment, and it would have been impossible to get such distribution by adding the entire 16 lb. at the coarse crusher.

The proper addition of lime to cyaniding ores becomes a difficult problem in the case of custom milling, where small lots of different ores are constantly going through the mill. Equal difficulty ensues in the case of ore from one mine in which the acidity fluctuates often. The most satisfactory solution for such problems is to bed the ores, thus making as homogeneous a mixture as possible. An alternative is to keep different classes of ore in separate bins and draw from each to make a uniform mixture.

Attention is called to the varying quality of lime, and to the necessity of buying lime on the basis of its "available" alkaline content. The test used for this purpose is that recommended by the Mines Trials Committee, of South Africa, and is as follows: To 2 gr. of lime crushed to 60-mesh, add 50 gr. of crystallized sugar and 800 c.c. of water. Shake thoroughly and allow to stand for a few hours with occasional shaking. Make up to 1000 c.c., shake and take out 50 or 100 c.c. with a pipette. Titrate with standard oxalic acid using phenolphthalein as an indicator. The standard oxalic acid can be prepared easily by dissolving 22.5 gr. of the crystallized acid in 1000 c.c. of water. In such a solution 1 c.c. = 0.01 gr. CaO.

Standardization of Metallurgical Terms.—Much confusion occurs in technical writing owing to the lack of uniformity in the use of technical terms. With a view to clarifying the situation, the Council of the Chemical, Metallurgical and Mining Society of South Africa has adopted certain standard terms and definitions for use in the society's *Journal*. Some of the terms refer mainly to Rand practice, but most of them will find acceptance in other countries.

1. The following words shall be treated as collective nouns, and therefore used in the singular, unless it shall be necessary to use the plural to make careful distinction: Sand, tailing, fine, black sand, residue, ash, pyrite, concentrate, slag, sulphide, telluride, slime, chloride.

2. The following shall be used in the plural form: Shavings, skimmings, steamings, sweepings, drillings, reserves.

3. Preference should be given to the singular form in: Precipitate, bin, cost, profit, value.

4. The following abbreviations shall not be printed in the plural: Oz., lb., dwt., gr. (grain), c.c., cm., kg., in., gal., m., gm. (gram), mm.

5. As far as possible the following terms (local expressions) shall be avoided: Dirt, mullock or muck (broken ore or waste), kokopan (truck), cheesa stick (firing stick), lasher (shoveler), lashing (shoveling).

6. The following spelling shall be used consistently: By-product, shoot (geological term), chute (opening through which material is guided), by-pass, hydraulicking, siphon, kafir, gram, dike, rise.

7. "Vat" shall be used instead of "tank," "cyaniding" instead of "cyanidation"; "breaking and sorting" instead of "crushing and picking"; (Note: "crushing" equals "stamp-milling" plus "tube-milling"); "ore slime" to distinguish from "gold slime"; "extraction" to refer to metal precipitated from solution; "recovery" to refer to metal won in a form suitable for market.

8. In referring to screening, it should be stated whether the number of holes refers to the square or linear inch, and the diameter of the aperture in decimals of an inch should be given; battery screen should have the gage of wire in decimals of an inch and the percentage of discharge area, mentioned whenever possible. Temperatures should be expressed in degrees Centigrade, with the equivalent in Fahrenheit in parenthesis. Cyanide and alkali strength shall be expressed in terms of specified reagents, thus: 0.05 per cent KCN; 0.02 per cent NaOH; 0.02 per cent CaO.

9. A ton shall be 2000 lb. A fluid ton shall mean 32 cu. ft.

10. The following scheme illustrates the nomenclature of Rand reduction works:

- Reduction works
  - Breaking plant
    - Stamp mill
      - Sets of stamp batteries
    - Tube mills or other secondary crushers
  - Crushing plant
  - Cyanide plant
    - Sand plant
    - Slime plant

Fuel.

Pulverized Coal for Metallurgical Furnaces.—It is only within the last 15 years that means for producing initially a fire of high efficiency have advanced to their present state of perfection. In a paper published in the October *Bulletin* of the A. I. M. E., Mr. H. R. BARNHURST presents a case for the use of pulverized coal in metallurgical furnaces The cement industry was the first in which pulverized coal was extensively used to supply heat for the chemical reactions taking place in the operations, but the production of heat in this way has since extended to other forms of chemical and metallurgical industry. Coal properly ground will burn thoroughly if 85 per cent passes through 200 mesh and 95 per cent through 100 mesh, the measuring screen apertures squaring 1/400 and 1/200 in. respectively. The author has found that such properly ground coal will contain a percentage above 70 capable of passing through a 300-mesh screen, squaring 1/600 in. on the side.

Undoubtedly the proper method of firing pulverized coal is to admit with the fuel the exact quantity of air necessary to the result to be attained, as shown by observation, and to maintain the relationship between the fuel and air quantities as long as conditions desired are being fulfilled. The control of the two factors, fuel and air, is and will be at the root of all success with pulverized or sprayed fuel in metallurgical processes. Pulverized fuel has a great advantage in that the eye of the intelligent operator can tell him whether he is getting the heat to serve his purpose. Also, it is best prepared for the absorption and evolution of heat in the combustion process, and in addition the air is properly prepared for joining in the process.

The author submits the following data on the use of pulverized coal in metallurgical furnaces: In roasting carbonate ores of high sulphur content, the carbon has been driven off and the sulphur reduced within permissible limits by the use of less than 7.5 per cent of fuel upon the weight of the charge. This problem involves the maintenance of a low temperature, about 2100 deg. F., to prevent agglomeration of the fine ore. The same practice obtains in the roasting and nodulizing of ores and flue dust, where the heat is sufficient to cause the material to form nodules but not stick to the furnace.

In open-hearth practice with pulverized coal, steel is being made with this fuel at the rate of from 450 to 500 lb. per net ton of product. This is an average of 45 heats, the fuel and product being carefully weighed. The melts were obtained in less time than with oil, which had been used previously. Following are analyses of slags with each fuel:

	Oil	Coal
SiO_2 per cent.....	16.0	16.5
FeO, per cent.....	22.0	18.2
MnO	7.4	6.7
P_2O_5	1.7	1.9

The sulphur in the final steel ranged from 0.025 to 0.035 per cent with oil, and 0.035 to 0.04 per cent with coal. The coal contained from 1 to 1.15 per cent sulphur.

In puddling furnaces it is safe to say that iron can be puddled at an average expense of 1200 lb. pulverized fuel per gross ton of muck bar produced; in fact, less than 1000 lb. of coal per gross ton of bars has been shown in practice during periods favorable to high economy.

The average consumption of pulverized fuel in heating furnaces seems to be from 500 to 550 lb. of fuel per gross ton. Busheling furnaces require from 550 to 600 lb.

"To obtain such results, however, the furnaces must be properly proportioned and equipped and in good condition. It must not be expected that the results obtained by simply squirting coal of greater or less degree of pulverization into a furnace, with an unmeasured jet of air, will equal the practice here shown. The figures given are authentic and may be relied upon. Success insists upon dry coal, fine pulverization, proper air supply. Another factor is that the attendants should be interested in the production of high results."

"With the proper predetermined proportioning of the apparatus, the operation will be elastic or adjustable to a wide range of performance under a very nearly constant percentage of efficiency. This is unattainable in an installation which may or may not be proportioned to the production of high efficiency at any particular duty. For this reason the ease with which pulverized fuel may simply be burned does not assure that the best results are being obtained."

Fuller's Earth

Bulletin No. 71 of the United States Bureau of Mines on the subject of "Fuller's Earth" has just been issued. Charles L. Parsons, the author, in the introduction says:

"The United States produces all of the fuller's earth used for refining petroleum within its borders. On the other hand, most of the fuller's earth used in bleaching edible oils has been imported from England. Recently a few of the refiners of cottonseed oil have begun to utilize domestic fuller's earth, whereas others have been unable to substitute it successfully for English earth in their practice. Many samples of American earth distinctly superior in bleaching power to the English earth have come to the attention of the Bureau of Mines. For these reasons an investigation of the mining, preparation, and use of fuller's earth in this country, especially in its application to edible oils, was conducted in order to ascertain why our own raw material has been deemed inapplicable to our needs.

"During the calendar year 1912, according to figures of the United States Geological Survey, there were imported into the United States 1,970 tons of unground fuller's earth, valued at $11,619, and 17,139 tons of ground fuller's earth, valued at $133,718, these values being based on the wholesale market price at the port of origin. The addition of transportation charges, commissions, etc., make the price to the American refiner $14.50 to $16 per ton. According to the latest figures compiled by the United States Geological Survey, the United States in 1912 produced 32,715 tons of fuller's earth, valued at $305,522 or $9.34 per ton, at the mine. Most of this domestic production was from three plants in Florida and one in southwestern Georgia and was used almost wholly for decolorizing petroleum. A small amount from other localities was used in the refining of edible oils. No figures definitely differentiating fuller's earth from other clays are kept in regard to our exports, but it is certain that several thousand tons of domestic earth was exported to Germany, and it is also true that German refiners of edible oils have used and are using large quantities of American fuller's earth and have a higher appreciation of its merits than our own refiners.

"As a result of the investigations made, the Bureau of Mines believes that the United States has fuller's earth far better suited for refining edible oils than any imported, and that to assure the almost universal use of this earth by American refiners there is required only a careful and intelligent technical control of the preparation of the output and its application to the bleaching of oils."

Copies of this bulletin may be obtained by writing to the Director, Bureau of Mines, Washington, D. C.

The International Oxygen Company announces that the steadily increasing demand for the I. O. C. system of oxygen and hydrogen generating plants has made necessary the removal of their executive offices from 115 Broadway, New York City to their works at Newark, N. J. Additional buildings have been erected to house the executive offices and provide additional manufacturing room for the increased demands made upon the plant. The general sales offices will remain at 115 Broadway, New York, as heretofore.

Dry Concentration and Separation of Minerals.

The Plumb Pneumatic Jig

(Editorial Correspondence)

Processes for concentration in the dry way have been experimented with for many years. The goal which has attracted inventors has been the evolution of a machine that would take the place of wet concentrators, particularly in localities where the water supply was insufficient to meet the requirements of wet ore-dressing. Some of the more recent proposals offer the possibility of direct competition with wet methods. The fact that only a few of the dry concentrators invented have ever attained commercial prominence, suggests the probability that the great majority reached only the patent-office stage of development.

Richards[1] classifies pneumatic concentrators under three heads: (1) those in which a continuous air-blast acts on the ore particles and yields graded products; (2) those in which the ore particles are projected through the air by some other force, and are graded according to their momentum; and (3) air jigs,

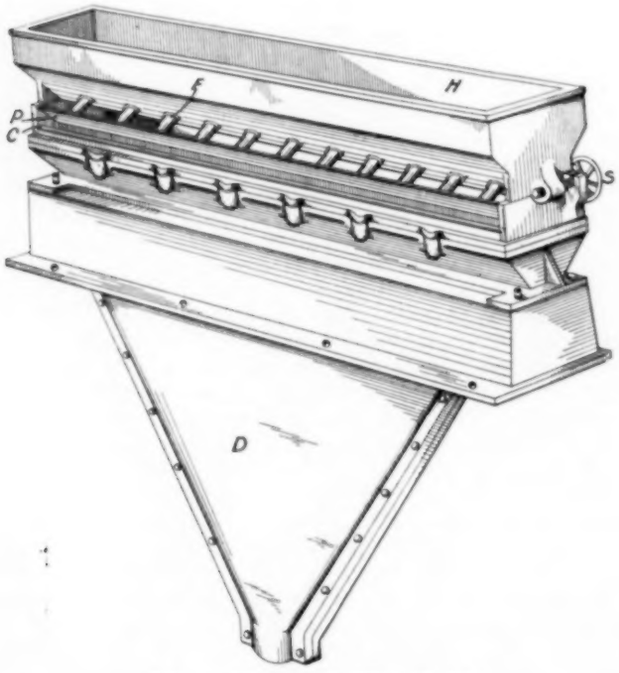


FIG. 1.—PERSPECTIVE VIEW OF JIG

or those in which a bed of ground ore is subjected to an upward pulsating current of air, resulting in a stratification of the constituent minerals and gangue according to their specific gravities. The pneumatic jig under consideration, which is the invention of Mr. A. M. Plumb, of Denver, Colo., falls naturally within the third class.

A suggestion which led to the development of the Plumb pneumatic jig was obtained from the operation of the Bonson dry concentrating table for iron ores. In this machine an adjustable deck is covered with two sets of riffle cleats or guides, one above the other, running diagonally to each other and at angles with the sides of the deck. When an ore mixture is fed onto the table, to which a reciprocating motion is imparted, a stratification of mineral and gangue occurs; and as the mass progresses over the deck a skimming action is produced by the upper set of cleats, whereby the lighter gangue is diverted in one direction while the concentrate is directed by the lower cleats to another point of discharge.

This process of skimming a stratified pulp by means of a guide or partition extending vertically downward into the mass, is one of the features of the Plumb jig. The method of securing stratification, however, is comparable to that used in hydraulic jigs, more particularly the Richards pulsator jig. A bed of dry ore on a screen is subjected to upward pulsations of air, without intermittent suction.

The construction and method of operation of this jig are best explained in connection with the accompanying drawings. Fig. 1 shows the jig in perspective, and Fig. 2 in cross-section.

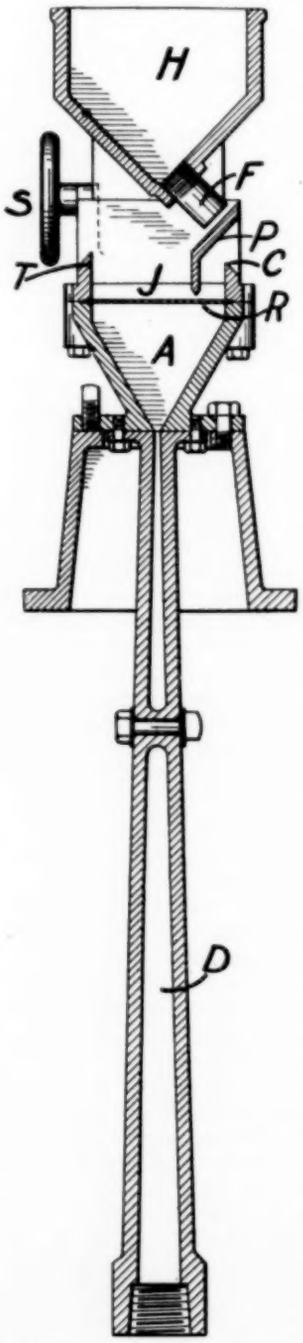


FIG. 2.—CROSS-SECTION OF JIG

An ore hopper H is provided with a number of feed pipes F, having their lower ends in proximity to the feed plate P. The hopper is adjustable with reference to the feed plate by means of screws S at each end. There is a screen at R on which the ore bed is supported. T is the line of tailing discharge, and C the line for delivery of concentrate. The jig bed J is divided by a vertical extension of the feed plate. There is an air chamber A below the screen, with sides converging to a narrow slot that registers with a slot of similar size in the triangular-shaped air distributor D. The latter is to be connected with a source of compressed air, and a rotary valve is placed in the air line near the jig.

The following dimensions have reference to a jig of commercial size. The base plate is 6 in. by 26 in. The inside dimensions of the jig bed J are 3 in. wide, 24 in. long and 1 in. deep, the last being the height above the screen of the tailing discharge. The concentrate compartment, separated from the main jig bed by the vertical partition, is 5/8 in. wide, 24 in. long and 0.87 in. deep, the last being the height of the concentrate discharge. The partition between the jig bed and concentrate compartment extends to within 3/16 in. of the screen. The 24-in. slot for admission of air is 1/16 in. wide. The screen is 150 mesh, supported between two 20-mesh screens. These dimensions may suggest a toy or model rather than a commercial concentrator, but the jig as outlined shows a capacity ranging from 5 to 10 tons per day, varying with the ore.

No less surprising is the fact that a single form of construction, with no adjustable parts save the feed hopper, should suffice for the treatment of practically all concentrating ores, both with regard to variations in the size of feed and mineral content. A fixed relation has been established between the heights of tailing and concentrate discharges, that will serve for the treatment of a wide range of ores. The conditions governing this fixed form of construction are given below, reference being had to Fig. 3. The figure represents a cross-section of the jig bed, drawn full size. When the jig is in normal operating condition on a given ore, the concentrate and tailing assume relative positions as shown. The tailing plus con-

[1] Ore Dressing, Vol. II, pp. 815-827, and Vol. III, pp. 1553-1555.

centrate in the jig bed will balance the concentrate in the other compartment.

Designate the heights of the tailing and concentrate strata in the jig bed as x and y, respectively, and the height of clean concentrate in the other compartment as unity. Assume an ore of the following characteristics:

Sp. gr. of the concentrate, $G = 5$
Sp. gr. of the tailing, $g = 3$

$$3x = 5y, \text{ or } x = \frac{5y}{3} \quad (1)$$

$$5x + 3y = 5 \quad (2)$$

Substituting in (2) the value of x in (1) we have

$$\frac{25y}{3} + 3y = 5, \text{ or } 25y + 9y = 15$$

Hence, $y = \frac{15}{25+9} = .441$. The general expression for the value y is $y = \frac{Gg}{G^2 + g^2}$

Substituting the value of y in (1) we get $x = \frac{2.205}{3} = .735$

Thus the height of $x + y$ for this particular ore will be 1.176, or 117.6 per cent of the height of the concentrate discharge.

For other combinations, for example where $G = 4$ and $g = 2.6$, or $G = 7$ and $g = 5$, the respective values of $x + y$ will be 1.157 and 1.137. The average of these conditions and of others likely to be encountered in concentrating ores will give a value of $x + y = 1.15$; so that if the height $x + y$ in the jig bed be made exactly 1 in., the height of the concentrate discharge will be 0.87 in., and these fixed dimensions will be within 2 per cent, plus or minus, of the figures indicated for any of the various combinations of mineral and gangue.

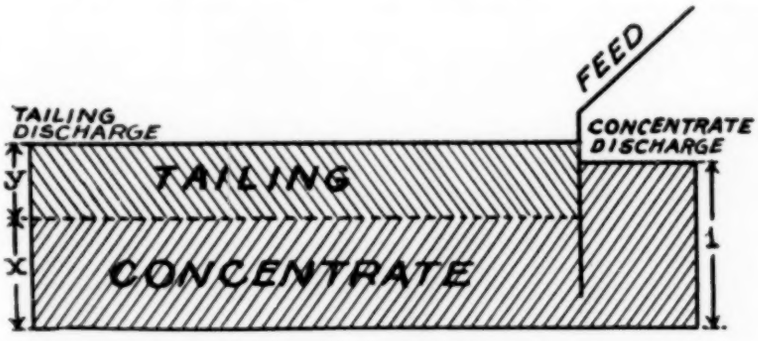


FIG. 3.—DIAGRAMMATIC CROSS-SECTION OF JIG BED

In operation, the dried ore pulp is fed through the hopper onto the screen. Under the action of the pulsating air current the ore bed becomes "light"; that is, the solid particles are separated from each other by the air which is rising through the mass and gradually expanding to atmospheric pressure. A condition of hindered settling in air is produced, which permits a ready stratification of the constituents of the ore, according to their specific gravities. After a few minutes of operation the bed acquires its normal operating condition for that particular pulp, and the jig bed assumes the condition shown diagrammatically in Fig. 3. The constant addition of further feed causes a displacement which results in the discharge of material over the two weirs, concentrate on one side and tailing on the other. The particles of heavy mineral pass down to the screen, flow under the partition, rise on the opposite side and ultimately pass out of the compartment due to the pulsating air. The tailing particles which are too light to sink into the concentrate stratum are constantly skimmed and discharged over the tailing weir. Thus a single machine can make but two products, concentrate and tailing. The latter may contain lighter minerals which will require a further separation.

Experimental work with the jig has resulted in defining very clearly its limitations as to size of feed. It is not well adapted to work on pulps coarser than 12 mesh, and it will not make a satisfactory separation of minerals ground finer than 150 mesh. Between these two limits, however, the machine gives satisfactory results.

It is a generally accepted principle that the feed for pneumatic concentrators should be more closely sized than for machines using water. The necessity for close sizing for the Plumb jig has not proved as great as might be generally supposed. The results of many tests indicate that three or four sizes between 12 and 150 mesh will be ample for good work, and that considerable latitude can be allowed in the sizing efficiency without materially altering the results of separation.

A convenient grading of the feed is $-12+20$, $-20+40$ and $-40+150$ mesh. Some data of operation are compiled in the following table:

Jig No.	Grade	Air Supply: Cu. ft. per min.	Air Supply: Pressure, lbs.	Air Supply: Pulsations per min.	Capacity, tons, 24 hr.
1	$-12+20$	40	60	700	10
2	$-20+40$	10	40	500	8
3	$-40+150$	3	15	500	4

Thus three jigs will have a combined capacity of about 1 ton an hour, for which the power requirement will be 8.6 hp.

One of the interesting and valuable features of the jig is its automatic regulation under variations of feed. For any given ore a maximum rate of feed will be determined, dependent on the quantity of mineral to be separated. For all variations in rate of feed between this maximum and zero, the jig is self-regulating and will perform its functions without adjustment. Should the feed cease entirely, the jig would merely stop discharging concentrate and tailing, and would continue to work indefinitely without disturbing the ore bed. When the feed came on again the separation would proceed as usual. An excessively high rate of feed would result in loss of mineral in the tailing.

Equally good regulation is obtained automatically when sudden changes occur in the grade of the feed. Thus if normal feed contains, say, 10 per cent galena, and a change should give 15 per cent, the line between concentrate and tailing (Fig. 3) would rise to a new position and remain there as long as the new feed continued. Should the grade fall to 5 per cent galena, the line between concentrate and tailing would be lowered. In the opinion of a competent authority this is the only jig ever built that perfectly regulates the discharge of concentrate and tailing.

The capacity of the jig will vary with the nature of the ore treated. It is limited by the rate at which concentrate can be discharged. Hence the tonnage capacity will be high on an ore with low mineral content, and *vice versa*. The jig shows its best capacity when treating an ore that yields three tons of tailing per ton of concentrate.

The field of usefulness of the jig is limited by the efficiency of coarse hydraulic jigs on the one hand, and slime concentrators on the other. In extremely favorable cases the jig might be the only concentrator needed to make the best commercial saving; but in most instances it will be used as an accessory to other concentrating and separating devices, such as tables and electrostatic separators. In any event, its work will be confined to that part of the ground or between 12 and 150 mesh in size.

The concentrates produced by this jig are always of high grade, regardless of the grade of the feed; that is, two ores containing 2 per cent and 10 per cent galena, respectively, will yield equally high grade concentrates. After the jig assumes normal operating conditions, as shown in Fig. 3, only heavy mineral particles can reach the screen and pass to the concentrate discharge. The production of high-grade concentrate is a great

advantage to the operator for it enables him to secure the highest market price for his products. In the case of a one-mineral separation, for example, the jig offers an opportunity to increase the quantity of high-grade concentrate, and thus produce a greater net profit, even if the total recovery is not increased. Improvement is shown, also, in the treatment of complex sulphide ores, where the total mineral recovery by wet processes usually is low. In some cases of this kind it has been possible by dry jigging and electrostatic treatment to separate and recover a part of the minerals at 12 mesh in the form of a clean, high-grade concentrate, leaving a tailing which contains the really complex part of the ore that requires fine grinding and separation on wet tables. Excellent results have been obtained in retreating concentrates from ordinary wet-dressing tables.

A few examples of the work of the jig are given below:

Retreating Products from Wet Concentration.—The two tests below show what can be done by dry jigging and electrostatic separation in raising the grade of products from table concentration. The original concentrate fairly represents the work done in wet-dressing a lead-zinc-iron ore. But there is a considerable quantity of zinc in the product which is without value; in fact, in the second case the zinc entails a penalty at lead smelters. By separating the original concentrate it is possible to secure this zinc in a merchantable product that will add to the net profit. The method of treatment in both of these tests was to jig the original table concentrate dry, getting a high-grade lead product and a tailing containing the zinc and iron with some lead. This tailing was then separated electrostatically on the Huff machine.

	Weight Per cent.	Lead Per cent.	Zinc Per cent.	Iron Per cent.
Test No. 1.				
Original table concentrate.....	100	54.50	10.71	9.85
Pneumatic jig concentrate.....	65	77.00	1.97	4.33
Pneumatic jig tailing, separated electrostatically into two products	20	19.80	8.68	14.00
	15	4.60	50.90	6.20
Test No. 2.				
Original table concentrate.....	100	40.86	14.70	10.30
Pneumatic jig concentrate.....	43	78.80	1.90	2.30
Pneumatic jig tailing, separated electrostatically into two products	31	18.75	5.60	26.40
	25	4.60	46.80	2.80

Treatment of Crude Ore.—In the following test the ore was crushed through 12-mesh and treated by pneumatic jigging, electrostatic separation and wet dressing, these different processes being applied to the portions of the ore best suited to the respective methods. The tables below show the weights and metal contents of ore and products, and percentages of metals recovered in different salable concentrates.

	Weight. Per cent.	Gold. Per oz.	Silver. Per ton.	Copper. Per cent.	Lead. Per cent.	Zinc. Per cent.	Iron. Per cent.
Crude ore	100.00	0.26	8.56	0.55	24.10	10.00	2.60
Pneumatic jig concentrate .	25.75	0.32	18.36	0.16	78.20	2.6	0.78
Electrostratic copper-iron .	5.89	1.09	36.90	5.55	31.18	13.20	11.50
Electrostratic zinc	14.00	0.14	3.62	0.34	5.36	48.09	1.69
Tailing	47.50	0.12	1.10	0.18	1.33	2.56	2.10
Dust and slime.	6.74	0.30	9.40	0.40	16.00	7.40	...

Percentage Recovery in Salable Products.

	Gold.	Silver.	Copper.	Lead.	Zinc.
Lead concentrates	31.70	55.25	....	83.56	....
Iron-copper concentrates...	25.50	25.30	59.5	7.60	....
Zinc concentrates	7.54	5.94	....	3.09	67.56
Total recovery.....	64.74	86.49	59.5	94.25	67.56

The pneumatic jig is not offered by its sponsors as a cure-all for the ills of concentration. Its apparent usefulness is within a well defined field, the limitations of which are appreciated. Developments up to date, however, indicate that even within its limited field the pneumatic jig can produce marked and favorable improvement in the present methods of concentrating and separating ores.

The jig is manufactured and sold by the Plum Jig Company, a subsidiary of the American Zinc, Lead and Smelting Company. The latter maintains a custom testing plant and assay laboratory at 1422 Blake Street, Denver, Col., where tests can be made on the Plumb jig, Huff electrostatic separator and other concentrating machines. The charges for the work are nominal considering the quality and extent of the tests.

Titanium for Rail Steel

The progress which metallographic methods have made in recent years in the metallurgical art is reflected in a characteristic way in three bulletins issued recently by the Titanium Alloy Manufacturing Company, of Niagara Falls, N. Y., and entitled "Rail Reports" Bulletins No. 1, 2, and 3.

A number of samples of standard and titanium-treated open-hearth rails were sent by several different railroad systems to the laboratories of the Titanium Alloy Manufacturing Company. All the samples were from "A" rails (the top rail from the ingot which usually shows the most excessive segregation); in each case both the standard and titanium-treated rolls had been rolled by the same mill, under the same specifications, with the single exception of the use of 0.10 per cent titanium in the treated steels.

The three bulletins give in detail the results of chemical and physical and microscopic tests of the seven different pairs of samples. A convenient summary of the results of the *chemical and physical* tests for the seven pairs of samples is given in Bulletin No. 3 in form of a long table from which the following general conclusions are drawn.

First. The tensile tests show on an average much better ductility for the treated rails, especially in the heads, and at the same time a slightly better strength than for the untreated.

Second. The impact tests show an average increase of shock resistance for the treated steel, in the head, web and flange, of about 50% over that for the untreated.

Third. The endurance tests on the White-Souther machine show that the treated rails are much less easily fractured by fatigue or by constantly repeated stresses below the elastic limit than the untreated.

Fourth. The chemical analyses, hardness tests, and Landgraf-Turner endurance tests (in which the stresses exceed the elastic limit), all show a much greater uniformity in the treated samples than in the untreated, indicating that the treated rails are freer from segregation and all its attendant evils.

To sum up all these points, it seems certain that these treated rails, when laid in the track and subjected to the strains and shocks of service, will form a safer line than would the untreated rails and that if excessive segregation is dangerous, the use of ferro-carbon-titanium in open-hearth rail steel is imperative.

Interesting as these physical and chemical tests are yet the beautiful series of sulphur prints and microphotographs given in the three bulletins are probably of even greater interest as they are a distinctly novel feature in a manufacturer's bulletin.

The *sulphur prints* are made by placing photographic paper on the polished cross sections of rail steels. This paper is moistened with a very weak sulphuric acid solution; and the sulphur in the steel reacts with the acid, forming hydrogen sulphide, which combines with the silver in the paper, producing dark spots thereon. The sulphur print is a rapid and accurate method of determining the quality of steel, and excessive segregation shown by this method indicates a weakness, which the detailed physical tests confirm.

Carbon, phosphorus, and sulphur segregate under similar conditions, so that the degree of segregation of sulphur as

shown by a sulphur print, determines the comparative degrees of segregation of phosphorus and carbon as well. Ample evidence is given in the latter of the three bulletins that this parallelism in the segregation of carbon, phosphorus, and sulphur is indeed a reality.

Cross-sections of the rails were also etched with iodine to reveal segregation and the pictures give, in the whole, the same indications as the sulphur prints.

Finally, a beautiful set of *photomicrographs* is given in each of the three bulletins. These photomicrographs were made of sections taken from the top of the head, center of the head, the web, the bottom of the flange, and the end of the flange. Further microphotographs were made of unetched sections as well as of sections under different etching treatments.

In short, these bulletins are exceedingly interesting. Quite apart from the primary object which is to permit the reader to form an unbiased judgment of the value of titanium treatment of rail steel they permit a very suggestive comparison between the different methods of testing—chemical, physical, and microscopic.

As to the value of titanium-treated steel rails in practice Messrs. Robert W. Hunt & Company are quoted in No. 3 as follows: "Through comparison of wearing results as between the standard open-hearth and the titanium-treated open-hearth rail, the latter in each of a number of instances on four railroad systems has demonstrated its superior wearing qualities. We have yet to find a single exception where the claim just pronounced is not true."

Wet and Dry Bulb Recording Hygrometer

Recording hair hygrometers, giving a single record of the relative humidity, have been in use for a long time. Engineers have appreciated the distinct advantage of having a record of both the dry-bulb temperature and the wet-bulb temperature independently but simultaneously on the same chart. Such an instrument has the added advantage in the ease with which its accuracy can readily be checked with a standard thermometer at all times. The importance of proper conditions of temperature and humidity is being more and more appreciated in its effect on health and human efficiency as also on the effect on many products during process of manufacture.

To meet the growing demand for a wet and dry bulb recording thermometer, the Industrial Instrument Company, Foxboro, Mass., have designed and placed on the market the recording hygrometer illustrated in Fig. 1. It consists of two sensitive bulbs mounted in tandem back of the case as illustrated in

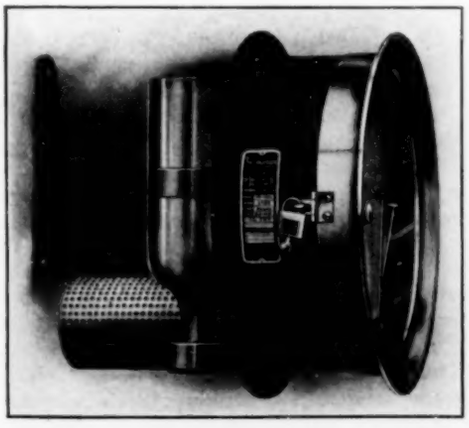


FIG. 1.—WET AND DRY BULB RECORDING HYGROMETER

Fig. 2, the wet bulb being jacketed and kept moist by maintaining water at a constant level in a trough beneath. The pen arms are attached directly to shafts concentric with the helical tube bulbs. The case is mounted on a swivel bracket enabling swinging the instrument at right angles to the wall or support, giving easy access to the inverted glass bottle serving as water reservoir. It is made in three sizes, viz.: 8 in., 10 in. and 12 in., corresponding to the standard sizes of Foxboro recorders and to cover ranges between freezing point and boiling point of water (32 deg. to 212 deg. Fahr., or 0-100 deg. C.).

Fig. 2 illustrates the manner in which the helical tubes acting

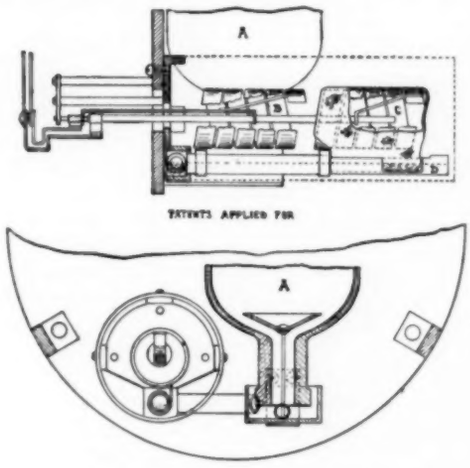


FIG. 2.—CONSTRUCTION OF HYGROMETER

as the bulbs are placed in relation to each other and the reservoir A.

These two bulbs extend beyond the case at the rear and the motion is transmitted to the pens through the actuating shafts. The tube marked B is the dry bulb which records the atmospheric temperature. The tube C is the wet bulb, which is covered with a special jacket, leading down into the trough *D*.

This bulb is always cooler than *B*, due to the evaporation of the water. This evaporation increases or diminishes in proportion to the amount of moisture in the air. Taking the difference between the two records made by the pens and consulting the table sent with the instrument, the relative humidity can be quickly ascertained.

Personal

Mr. G. F. Brindley, who was works manager of the Niagara Electrochemical Company, Niagara Falls, N. Y., from 1896 to 1906, when he left for Japan, subsequently held the position of consulting chemist and engineer to Dr. F. S. Pearson in Mexico, and has recently returned from a prolonged stay in New Brunswick where he was carrying on an investigation of the oil shales of that place. He is at present doing some experimental work at The FitzGerald and Bennie Laboratories, Niagara Falls, N. Y.

Mr. Franz Cazin, of Denver, Col., has been obliged to return to the United States from Mexico, where he was engaged in the design and construction of a large metallurgical plant. Conditions in the southern republic made further work impossible.

Mr. Robert C. Gemmel, general manager for the Utah Copper Company, celebrated his twenty-fifth wedding anniversary on October 17, at which time the operating heads of the Utah Copper Co. presented Mr. and Mrs. Gemmel with a beautiful silver punch service. The bowl is engraved with a panorama of the great Utah copper mine and with scenes depicting the mining and transportation of ore.

Mr. Jay P. Graves, who has been vice-president and general manager of the Granby Consolidated Mining, Smelting & Power Co., of British Columbia, has retired from the active management of the company. He is succeeded by **Mr. F. M. Sylvester,** who has been assistant general manager.

Mr. John Gross, metallurgical engineer of Denver, Col., has finished the construction of a new mill for the London Mines Company, in Gilpin county.

Mr. John T. Jones, of Iron Mountain, Mich., was in Denver recently en route to Utah and Arizona on professional business.

Mr. Robert M. Keeney, of the Bureau of Mines, Pittsburgh, Pa., was married October 22 to Miss Maud Conrad. Mr. and Mrs. Keeney are at home at the Moyer apartments, Oakmont, Pa.

Mr. Thomas M. Kekich has been engaged by the Spassky Copper Mines, Ltd., Siberia. He recently resigned his position as assistant superintendent of the Great Western S. & R. Company, Chicago.

Sir Robert Mond and **Sir Alfred Mond** were the guests of honor at a dinner tendered by Dr. L. H. Baekeland at the Chemists' Club in New York before their return to England. Dr. Baekeland acted as toastmaster. Sir Robert Mond spoke impressively of the scientific, industrial and commercial problems inherited from his father. Sir Alfred Mond, M. P., spoke very amusingly of the life of a British politician. The other speakers were Dr. W. H. Nichols, Dr. C. F. Chandler and Dr. J. Takamine.

Mr. H. N. Thompson has resigned his position as superintendent of the Tooele smelter of the International Smelting & Refining Co., in Utah. **William Wraith**, manager of the plant, will assume the duties of Mr. Thompson, and will be assisted by **O. M. Ruchs**, who has been chief chemist.

Obituary

Ashmead Gray Rodgers for twelve years superintendent of The Carborundum Company's plant at Niagara Falls, died October 23d, 1913, as the result of injuries sustained through an accident on October 5. Mr. Rodgers had a host of friends and acquaintances throughout the mechanical and chemical world and they will sincerely feel the loss of a man who was so pleasing in personality, democratic in his attitude to others, and so capable in his chosen line of work. Mr. Rodgers was a native of Albany, N. Y., having been born there in 1872. Previous to his coming to The Carborundum Company as superintendent he was superintendent of the Eddy Electrical Company, of Hartford, Conn. His funeral services were held Saturday, October 25, from St. Peter's Church at Niagara Falls and was attended by several hundred of the employees of The Carborundum Company, members of the Niagara Club and other friends. Mr. Rodgers was a member of the American Electrochemical Society, the American Chemical Society, Engineers' Society of New York, Chemists' Club, Niagara Club, University Club and Country Club of Niagara Falls and several other scientific and social organizations.

Book Reviews

Metallography. By **Cecil Desch.**

This book was reviewed in our November number, but the title erroneously given as "Metallurgy."

New Standard Dictionary of the English Language. Edited by Isaac K. Funk; prepared by nearly four hundred specialists. Large quarto (30 x 22 cm.), xxxviii + 2916 pages, nearly 10,000 illustrations, 450,000 definitions. Price, $21.50, in better bindings, $27 and $35. New York and London: Funk & Wagnalls Company.

The appearance of a new complete dictionary, prepared with the great expenditure of money and brains which have been lavished on this new edition of the Standard, is an event worthy of notice in a technical journal. Confining our remarks to the scientific side of the work, and particularly to that which concerns our readers, chemical terms have been supervised by Dr. H. W. Wiley, geology by Director George Otis Smith, of the U. S. Geological Survey, metallurgy and electrochemistry, by Dr. J. W. Richards, mineralogy, by Dr. F. W. Clarke, gems and precious stones by Dr. G. F. Kunz, electricity by Dr. Samuel Sheldon, engineering and mechanical terms, by C. A. Munn, mining by Dr. J. C. Bayles, former editor of *The Iron Age*. With equally competent experts in charge of other departments of the book, the result is probably as near perfection as it is humanly possible to make such a colossal work. From the printer's and publisher's standpoint, it is a masterpiece of the book-maker's art.

General Metallurgy. By **H. O. Hofman**, Professor of Metallurgy in the Massachusetts Institute of Technology. Large octavo (15 x 23 cm.), 909 pages, 836 illustrations; price $6.00 net. New York: McGraw-Hill Book Company, Inc.

This remarkable book is the equivalent of Schnabel's "Allgemeine Hüttenkunde," brought up to date and written in the spirit of modern American metallurgy.

A rather short chapter on the properties of metals is followed by a longer one on alloys; metallic compounds are then described in general terms, with their heats of formation. Over 300 pages are given to "Fuel"; a short 38-page chapter to refractory materials, and 124 pages to smelting processes in general, including fluxes and slags. Only 8 pages are given to the details of hydrometallurgical processes, and but 20 pages to electrometallurgical methods and apparatus. On the other hand, an imposing chapter of 350 pages covers mechanical operations, such as handling ores, working metals and alloys, pumping and filtering liquids, supplying, heating and drying blast, purification of gases and catching of fume.

There is an extraordinary amount of useful information in this work, rendered doubly valuable by numerous references to the literature, so that the reader is everywhere informed where further information can be obtained. There are also numerous mistakes, such as are perhaps unavoidable in compiling such an enormous work, but which must be corrected in the next edition. The reviewer particularly calls attention to the chapter on "Fuel," in which small calories (cal.) and large calories (Cal.) are frequently confused, and in which flame temperatures are in some cases evaluated on a false principle. These, and rather numerous smaller errors and typographical mistakes, are blemishes on the usefulness of what is otherwise a magnificent work and a credit to American technical literature; assuming these corrected in the next edition, we will then be able to recommend the work to students and practitioners, without reservation, as just the work on general metallurgy that we have for so long desired to have.

J. W. Richards.

Industrial Poisoning, from Fumes, Gases and Poisons of Manufacturing Processes. By Dr. **J. Rambousek,** Professor of Factory Hygiene and Chief Health Officer of Bohemia. Translated and edited by **Thos. M. Legge,** M.D., Medical Inspector of Factories. Octavo (21 x 13½ cm.), 360 pages, 61 illustrations. Price, $3.50 net. New York: Longmans, Green & Co. London: Edward Arnold.

The author is a medical man, a chemist, and a government official, and has treated the subject in a novel, comprehensive and systematic manner. The translator is a physician, and lecturer on factory hygiene in the University of Manchester, England.

The first part of the book describes briefly the process of manufacture of metals and chemicals, the opportunities for poisoning in each case and the symptoms and effects. Part two deals with industrial poisoning in general, its classification, pathology and treatment. Part three deals with preventive measures in general, and in particular cases. A classified bibliography is given in an appendix. One omission is the peculiar poisoning caused by the dust of vanadium ore and oxide, but this is a rather recent discovery; another is the dangerous effects of osmium oxide vapor. The book is, however, very complete, and so well written as to be really fascinating in its interest.

Every works or firm manufacturing, handling or using chemicals or metals should get a copy of this book.

METALLURGICAL AND CHEMICAL ENGINEERING

A Monthly Journal being the Incorporation of
Electrochemical and Metallurgical Industry
and Iron and Steel Magazine.

VOLUME XI

From January to December 1913

NEW YORK:
McGRAW PUBLISHING CO.